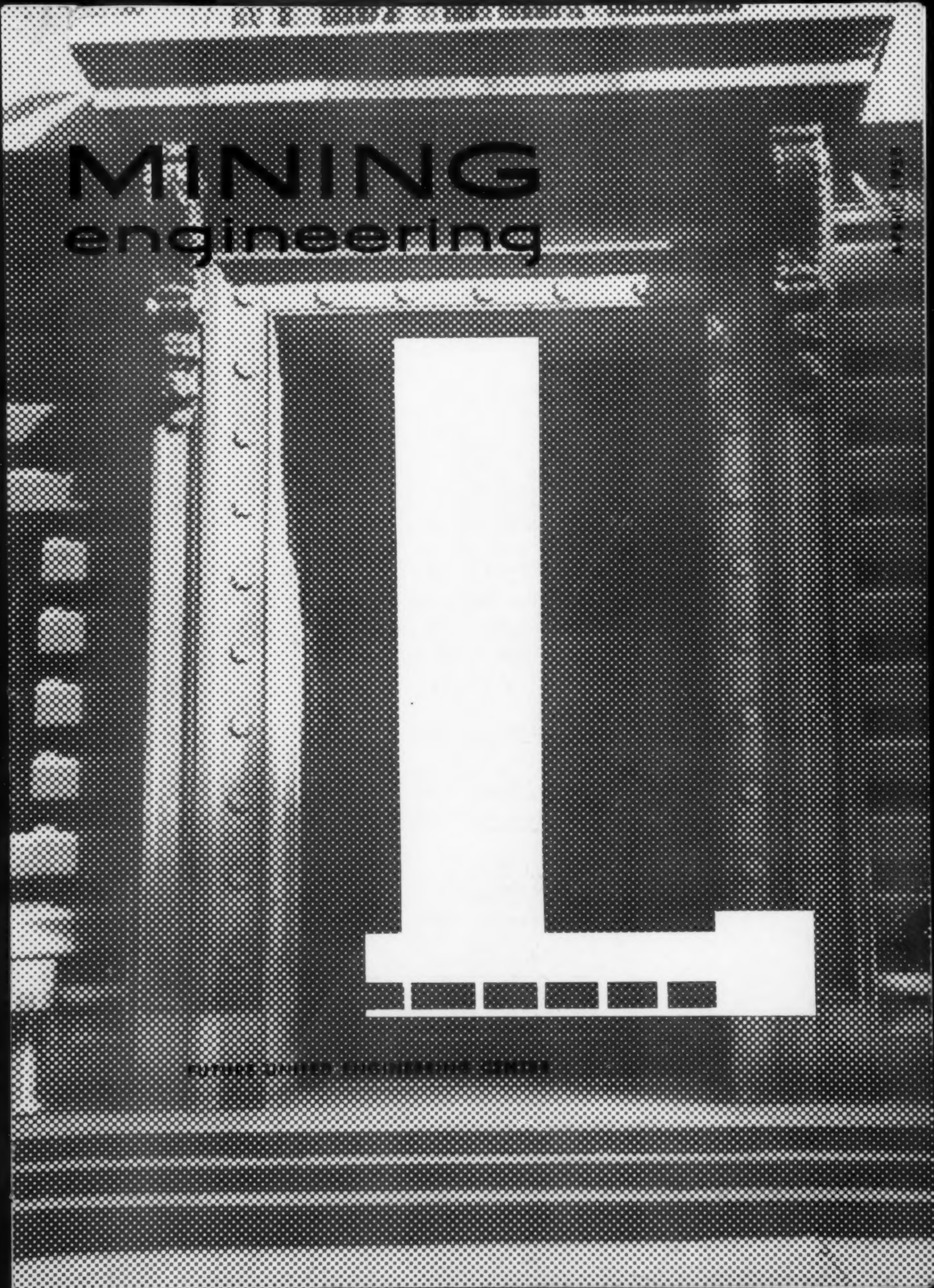
MINING engineering

APRIL 195[illegible]



FUTURE UNITED ENGINEERING CENTER

Coming Events

Apr. 11, AIME Reno Subsection, 12 noon, Nevada Room, Hotel Mapes, Reno, Nev.

Apr. 14-15, American Zinc Inst., Chase-Park Plaza Hotels, St. Louis.

Apr. 15, AIME Utah Section, joint meeting with Student Chapter, University of Utah; J. L. Gillson, AIME Vice President, speaker; University of Utah, Salt Lake City.

Apr. 15-16, Lead Industries Assn., Chase-Park Plaza Hotels, St. Louis.

Apr. 17-19, AIME Pacific Northwest Regional Conference, Spokane.

Apr. 18-20, AIME Alaska, Southwest Alaska Sections, mining and minerals conference, School of Mines, University of Alaska, College, Alaska.

Apr. 21-22, Inst. on Lake Superior Geology, University of Minnesota, Duluth Branch, Duluth.

Apr. 23-24, Bituminous Coal Research Inc., annual meeting, Penn-Sheraton Hotel, Pittsburgh.

Apr. 25, AIME Pennsylvania-Anthracite Section, spring technical meeting, Hazelton, Pa.

Apr. 26, AIME Adirondack Section, St. Lawrence Power Project and Seaway tour; dinner meeting; J. W. Woomer, SME President-Elect speaker; Village Inn, Massena, N. Y.

May 5-7, American Mining Congress Coal Convention, Netherland Hilton Hotel, Cincinnati.

May 8, AIME Utah Section, W. R. Hibbard, Jr., speaker; subject: Alloy Developments, Salt Lake City.

May 9, AIME Reno Subsection, Nevada Room, Mapes Hotel, Reno, Nev.

May 9-10, Dept. of Mining Engineering, Montana School of Mines and AIME, Mining Assn. of Montana, The Anaconda Co., Montana Soc. of Engineers, symposium on hydraulic emplacement of mine stope fill, Montana School of Mines, Butte, Mont.

May 9-11, AIME Uranium Section, Third Annual Uranium Symposium, Moab, Utah.

May 22-23, 34th annual conference, Lake Superior Mine Safety Council, Hotel Duluth, Duluth.

May 23, AIME Lehigh Valley Section, dinner-dance, Saucon Valley Country Club, Bethlehem.

May 24, AIME Colorado MBD Subsection, Broadmoor Hotel, Colorado Springs, Colo.

June 13-14, AIME Central Appalachian Section and SME Coal Division, joint meeting, Phoenix Hotel, Lexington, Ky.

June 22-25, Annual Convention of the Mine Inspectors' Inst. of America, Shirley-Savoy Hotel, Denver.

June 26, AIME Pennsylvania-Anthracite Section, summer meeting, Split Rock Lodge, White Haven, Pa.

Sept. 17-19, AIME Rocky Mountain Minerals Conference, Newhouse Hotel, Salt Lake City.

Sept. 22-25, American Mining Congress Mining Show, Civic Auditorium, San Francisco.

Oct. 2-4, Annual Drilling Symposium, University of Minnesota, Minneapolis.

Oct. 9-10, AIME-ASME Solid Fuels Conference, Hotel Chamberlin, Old Point Comfort, Virginia.

Oct. 23-25, AIME Mid-America Minerals Conference, Chase-Park Plaza Hotels, St. Louis.

Oct. 29-Nov. 1, Soc. of Exploration Geophysicists, annual meeting, Roosevelt Hotel, New Orleans.

Feb. 15-19, 1959, AIME Annual Meeting, San Francisco.

MINING engineering

VOL. 10 NO. 4 APRIL 1958

COVER

Silhouetting the *New* against the *Old*—for this month's cover artist Herb McClure has chosen to highlight progress being made toward completion of the United Engineering Center. For a resume of events to-date, see Drift, page 449.

ARTICLES

TRANSACTIONS

FEATURES

Published monthly by the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., 29 West 39th St., New York 18, N. Y Telephone: PEnnsylvania 6-9220. Subscription $8 per year for non-AIME members in the U. S., & North, South, & Central America; $10 foreign; $6 for AIME members, or $4 additional for members only in combination with a subscription to "Journal of Metals" or "Journal of Petroleum Technology". Single copies, $.75; single copies foreign, $1.00; special issues, $1.50. The AIME is not responsible for any statement made or opinion expressed in its publications. Registered cable address, AIME, New York. Indexed in Engineering Index, Industrial Arts Index, and by The National Research Bureau. Second class mail privileges authorized at New York, N. Y., and additional entry established at Manchester, N. H Number of copies of this issue 15,300.

PERSONNEL

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc. (Agency) operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

— MEN AVAILABLE —

Geologist, B.A., age 37. Three years geologist and chief engineer large tin mine handling mapping, surveying, development, and stope projects, sampling ore reserves. Five years in charge of exploration programs using all modern methods. Prefer Latin America, foreign, west. M-400.

Mining Engineer, Dutch, 36, married, graduate Technological University of Delft (Holland). Eight years experience exploring and operating tin and tungsten mines in Central Africa, surface and underground. Desires position as mining engineer or geologist. Location, foreign. Fluent French, German, and English. M-402.

Production Supervisor, B.S. in mining engineering, age 31. Five years professional experience open pit, underground metal mines in western U. S. and South America. Two years practical mining experience western metal mines. Speak Spanish. Will accept position in southern U. S. or Latin America if there are facilities for family. M-403.

Mining Engineer, B.S. in mining engineering, age 31. Eighteen months mine foreman, underground copper mine; six months mine surveying, underground; one year assayer for Nevada Bureau of Mines. Prefer U. S. M-404-1016-San Francisco.

Mining Engineer-Geologist, 48, excellent health, no physical defects, married, one child, would consider single status. Twenty-five years experience all types underground mining, good production cost record. Can handle men. Experienced geologic examinations, evaluation reports. Ample executive experience and ability. Presently employed as manager South American company. Available reasonable notice. M-405.

Geologist or Junior Engineer, A.B., University of California, age 28. Two years petroleum geologist, one year field geologist and mining engineer, two years many phases of seismic exploration. Prefer U. S. S-1038.

Sales Engineer, B.S. in mining engineering, age 34. Three years selling complete line of mining equipment and machinery, both air and hydraulic electrically operated; two years selling electrical motors and drives. Location, immaterial. M-406.

— POSITIONS OPEN —

Mine Superintendent, coal mine operations, to take responsible charge of design, construction, mine development, maintenance, and operation of large capacity modern mines. Excellent living conditions; rapid advancement. Apply by letter giving experience, personal status, and salary expected. Location, South. W5916.

General Manager, graduate mining engineer, to take complete charge of an open pit mining company mining phosphate rock. Will have direct supervision of mining, processing, sales, economics, etc., reporting directly to president. Salary, $25,000 to $30,000 a year. Location, East. W5910.

Research Metallurgist, to 45, metallurgist or mining engineer, well trained in metallurgy, with aptitude and interest primarily research, to improve current practice in 100-tph copper flotation mill and test various grades and mixtures of massive sulfide ores. Three-year contract. Salary, open. Submit complete record and references with first reply. Location, Mediterranean area. F5884-S.

General Manager, engineering graduate, with underground mine supervision, milling, and salt refining experience, to take complete charge of salt property. Salary, $10,000 a year, plus car, housing, and bonus. Location, Caribbean area. F5505.

Sales Engineer, young, wtih engineering training and some mining experience, for sales and service work covering rock drills. Salary, $6000 a year plus bonus. Location, New York. W5417.

Mine Superintendent, mining engineering graduate, with about ten years experience in Latin America. Salary, $10,000 a year plus bonus and housing. Location, South America. F-4603.

BOOKS

Better Wage Incentives, by Phil Carroll, *McGraw-Hill Book Co. Inc.,* 222 pp., $4.75, 1957.—Guide to setting up and maintaining an incentive program, covering various types of incentive grievances, how to plan to prevent them, and what to do if they arise. •

Mining in Canada, *Sixth Commonwealth Mining and Metallurgical Congress,* 507-837 W. Hastings St., Vancouver, B. C., Canada, 597 pp., $10, 1957.—Volume is a joint effort by the Metal Mining and Coal Divs., CIM, and prepared for the visit to Canada of the Commonwealth Congress. Articles were compiled by International Nickel Co. staff members and cover all phases of the industry in Canada. •

Guide to Career Information, by the New York Life Insurance Co., *Harper & Brothers,* 203 pp., $3, 1957.—A sourcebook of occupational information, it lists over 800 books and pamphlets devoted to current job information, grouped under 52 occupational categories. After each listing is given a brief resume, including price, number of pages, and where material may be obtained. •

Coal-Mining Economics, Organization and Management, by John Sinclair, *Sir Isaac Pitman and Sons Ltd.,* London, England, 337 pp., approx. $7.50, 1957.—Beginning with a brief outline of the growth of the British coal mining industry and an overall view of its financial structure, the book proceeds to a study of management and organization in relation to the techniques of work and study method, joint consultation and conciliation, trade unions, and various aspects of human relations. Also included is the organization of finance, budgetary control and costs, valuation, and plant maintenance. Marketing and transportation arrangements are discussed. •

Radioactive Mineral Occurrences in the Bancroft Area, by J. Satterly, Vol. 65, *Ontario Dept. of Mines,* Ottawa, Ont., Canada, 50¢, 1958.—The story of the feverish prospecting and mining activity in the Bancroft area of southeastern Ontario is incorporated in Vol. 65 of the department's annual report. Descriptions are given of about 125 properties that were subjected to exploration. An overall assessment of the geology of the area is given.

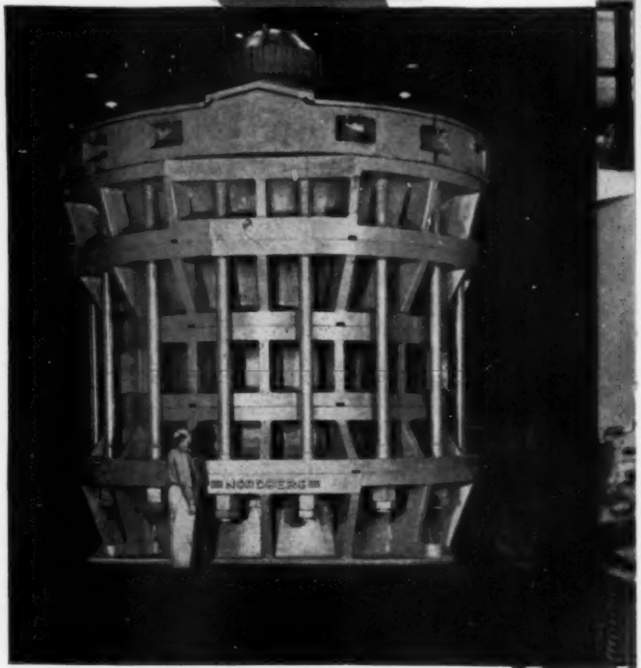
Appraisal and Valuation Manual, *American Soc. of Appraisers,* Manual Div., 369 Lexington Ave., Suite 1105, New York 19, N. Y., 500 pp., $15, 1958.—Source book of latest information on the solution of appraisal and valuation problems encountered in government and business. •

Qualitative Testing and Inorganic Chemistry, by Joseph Nordmann, *John Wiley & Sons Inc.,* 476 pp., $6.25, 1957.—The book contains numerous problems in calculations worked out within the chapters; practical applications of the subject to industrial chemistry; chemistry and analysis in common alloys; treatment of elementary chemical equilibrium; organic analytical reagents; review of equation writing, mathematical manipulations, and expressions of solution concentrations; and sections on blow pipe analysis, dry fusion, and bead and flame tests. •

Scientific Societies in the United States, by Ralph S. Bates, 2nd edition, *Columbia University Press,* 2960 Broadway, New York 27, N. Y., 268 pp., $6.50, 1958.—Account of the development and influence of scientific societies in the U. S. begins with the 18th century. The author, in this new edition, examines the recent strengthening of the ties between various societies and the work being performed by UNESCO and other international organizations. •

(*Continued on page 429*)

NORDBERG MACHINERY
... used by leading TACONITE producers



This Symons Primary Gyratory Crusher has a feed opening of 54" and capacity of 2000 or more tons per hour of hard, abrasive Taconite ore. Symons Gyratories are built in sizes up to 72" feed openings, with capacities of over 3500 tons per hour.

To provide adequate iron ore reserves for today's increased steel production, the large-scale beneficiation of Taconite ore has become imperative.

Long used in the economical production of Taconite concentrates in the Arctic region of Norway, Nordberg Machinery is now serving the huge Taconite operations in northern Minnesota . . . where a new era in U.S. iron ore production is now underway.

The selection of Nordberg Machinery for the processing of the extremely hard Taconite ore again follows the trend of the mining fraternity to rely on this heavy duty machinery for its proven ability to reduce ores and minerals at low per ton cost.

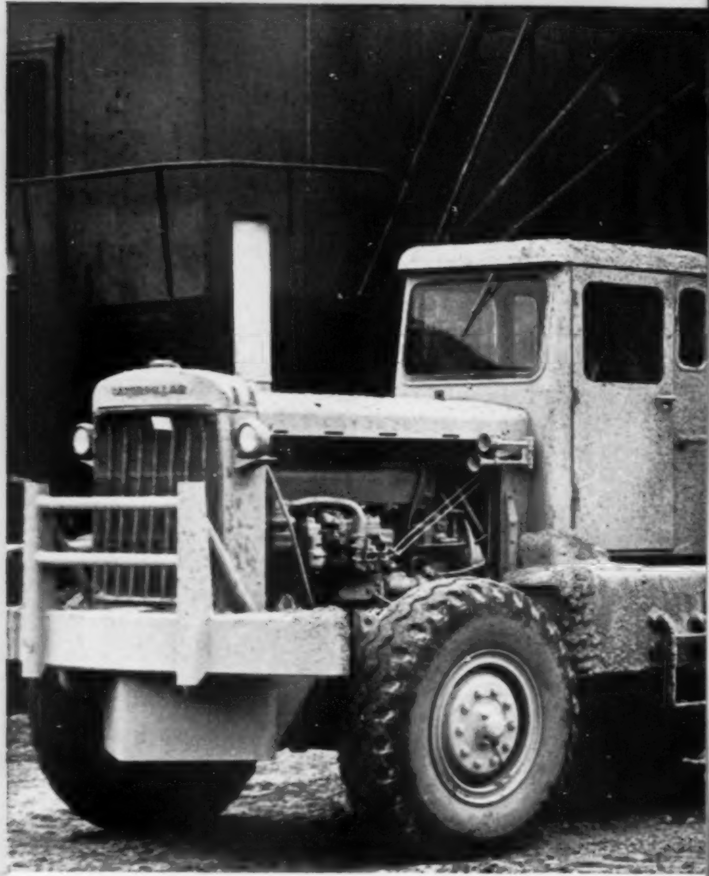
Nordberg Machinery now in use by leading Taconite producers includes Symons® Primary Gyratory Crushers, Symons Cone Crushers, Nordberg Rod and Ball Mills, and Symons Rod Deck Screens.

While *your* crushing and processing problems may not involve Taconite, you may be sure that Nordberg machinery is the logical choice for practically any other ore and mineral processing operation where big tonnages at low cost are required.

NORDBERG MFG. CO., MILWAUKEE,

Hauls Material . . .

An Athey First.. "T-1" Steel in Bottom Dump Trailer Design



6 New DW20-PW20 Bottom Dumps

Utah Construction Company's fleet of 6 PW20 Bottom Dumps has been stripping overburden from the copper deposits at a large open-cut copper operation. The new Athey-Cat PW20-DW20 Bottom Dump Trailers were selected on the basis of their ability to produce. Utah reports, "The units are giving good performance. We like their low vehicle weight-to-load ratio."

The Athey PW20 is *all new*, with many design features for faster, lower cost hauling. Dead weight is trimmed to minimum by the use of high-strength "T-1" steel. The DW20-PW20 team handles 40-ton payloads – nearly 4 times the trailer weight.

Three doors – instead of *two* – dump the load. When the two extra-wide bottom doors open, the rear door swings back – ends "hang up" on the dump.

Books

(*Continued from page 420*)

Economics of American Industry, by Evan B. Alderfer and H. E. Michl, 3rd edition, *McGraw-Hill Book Co. Inc.*, 710 pp., $7, 1957.—Revised to incorporate the most recent changes, this edition analyzes the economics and structures of the major American industries. All new materials are analyzed with respect to the changing competitive pattern in each industry and the competitive realignments of all manufacturing, i.e., the interindustry competitive. •

Making a Work Sampling Study, film produced by *The University of California*, Educational Film Sales Dept., Los Angeles 24, Calif., 16-mm, sound, color or black and white, 23 min., $195 color, $110 black and white, 1957.—The film opens with the statement of a manufacturing problem and then shows, step by step, the application of work sampling as the means of obtaining the facts to enable solution of the problem.

Bibliography of Graduate Theses on Geophysics in U. S. and Canadian Institutions, by George E. Tarbox, Vol. 53, No. 1, Quarterly of the *Colorado School of Mines*, Golden, Colo., 55 pp., $1, 1958.—Pamphlet contains list of theses by subect, by school of origin, and by authors.

State Publications

Minerals of California, Bulletin 173, by Joseph Murdoch and Robert W. Webb, *State of California Div. of Mines*, Ferry Building, San Francisco 11, Calif., 452 pp., $3 plus tax, 1958.—Eighth of a series, the book is essentially a catalog of the minerals found in California. About 523 minerals are listed, alphabetically, with data on the occurrence of each tabulated by county. Approximately 2000 references are given.

Mineral Commodities of California: Geologic Occurrence, Economic Development, and Utilization of the State's Mineral Resources, staff written, Bulletin 176, *State of California Div. of Mines*, Ferry Bldg., San Francisco 11, Calif., over 700 pp., $7.50 plus tax, 1958.

Mineral Industries Bulletin, bimonthly, *Colorado School of Mines* and *Colorado School of Mines Research Foundation*, Golden, Colo., available free upon request. First issue was on beryllium by Donald R. Williamson. March issue is on lithium and its uses.

Precambrian and Tertiary Geology of Las Tablas Quadrangle, New Mexico, by Fred Barker, Bulletin 45, *State Bureau of Mines and Mineral Resources* and *New Mexico Institute of Mining & Technology*, Campus Station, Socorro, N. M., 97 pp., 14 tables, 3 figs., 13 plates, including 3 maps, $2.75, 1958.

Geological Investigations, map covering 200 sq miles in the southwest quarter of the Philipsburg quadrangle, by Glenn J. Poulter, *Montana Bureau of Mines and Geology*, Room 203-B, Main Hall, Montana School of Mines, Butte, Mont., 28x42-in. map, text sheet, 40¢, 1958.

Preliminary Report on the Sedimentary Uranium Occurrences in the State of Pennsylvania, by John F. McCauley, Progress Report 152, *Bureau of Topographic and Geologic Survey*, Dept. of Internal Affairs, State of Pennsylvania, Harrisburg, Pa., free upon request, 1958.

Refraction Seismic Investigations: Rosiclare Fluorspar District, Illinois —Part I. Goose Creek Area, by Robert B. Johnson, Circular 231, *Illinois State Geological Survey*, Urbana, Ill., 15 pp., 7 figs., 2¢ postage, 1957.

Groundwater Geology in Western Illinois, South Part: A Preliminary Geologic Report, by Robert E. Bergstrom and Arthur J. Zeizel, Circular 232, *Illinois State Geological Survey*, Urbana, Ill., 28 pp., 7 figs., 1 table, 3¢ postage, 1957.

Pottery Clay Resources of Illinois, by Edward C. Jonas, Circular 233, *Illinois State Geological Survey*, Urbana, Ill., 8 pp., 2 figs., 2¢ postage, 1957.

Microscopy of the Resin Rodlets of Illinois Coal, by Robert M. Kosanke and John A. Harrison, Circular 234, *Illinois State Geological Survey*, Urbana, Ill., 14 pp., 2 plates, 3¢ postage, 1957.

Gypsum and Anhydrite in Illinois, by Donald B. Saxby and J. E. Lamar, Circular 226, *Illinois State Geological Survey*, Urbana, Ill., 26 pp., 7 figs., appendix, 3¢ postage, 1957.

Weathering of Illinois Coals During Storage, by H. W. Jackman, R. L. Eissler, and F. H. Reed, Circular 227, *Illinois State Geological Survey*, Urbana, Ill., 22 pp., 5 figs., 18 tables, 2¢ postage, 1957.

Subsurface Dolomite and Limestone Resources of Grundy and Kendall Counties, by Meredith E. Ostrom, Circular 230, *Illinois State Geological Survey*, Urbana, Ill., 25 pp., 7 figs., 1 table, 3¢ postage, 1957.

U. S. Bureau of Mines Publications

Request free publications from:
Publications Distribution Section
U. S. Bureau of Mines
4800 Forbes Street
Pittsburgh 13, Pa.

RI 5366 Laboratory Investigation of Bauxite Ore from the Quapaw Deposit, Saline County, Ark.

RI 5367 Field-Scale Experiments in Underground Gasification of Coal at Gorgas, Ala.: Use of Electrolinking-Carbonization as a Means of Site Preparation.

IC 7806 Use of Lignin Sulfonate for Dust Control on Haulage Roads in Arid Regions.

IC 7807 Mining and Milling Methods and Costs, Ozark Ore Co. Iron Mountain Iron-Ore Mine, St. Francois County, Mo.

IC 7809 Making Ventilation-Pressure Surveys With Altimeters.

IC 7810 Recommended Safety Standards for Shaft Sinking.

IC 7811 Mining Methods and Costs, Calyx Nos. 3 and 8 Uranium Mines, Temple Mountain District, Emery County, Utah.

IC 7812 Thickness of Bituminous-Coal and Lignite Seams at all Mines and Thickness of Overburden at Strip Mines in the United States in 1955.

U. S. Bureau of Mines Publications

Copies sold through:
Superintendent of Documents
Government Printing Office
Washington 25, D. C.

Bulletin 562 Mine Flood Prevention and Control: Anthracite Region of Pennsylvania. Final Report of the Anthracite Flood-Prevention Project Engineers, 60¢.

Bulletin 573 Micro-Seismic Method of Determining the Stability of Underground Openings, 35¢.

Miners' Circular 52 Accidents From Falls of Rock or Ore at Metal and Nonmetallic Mines. Metal and Nonmetallic-Mine Accident-Prevention Course—Section 2, revised July 1955, 40¢.

IC 7805 Bibliography on Extractive Metallurgy of Nickel and Cobalt, January 1929–July 1955, 70¢.

Annual Lists of Publications: 1956, 1955, 1954, *U. S. Bureau of Mines*, Publications-Distribution Section, 4800 Forbes St., Pittsburgh 13, Pa., or Interior Bldg., Washington 25, D. C., gratis.

U. S. Bureau of Mines Comprehensive Catalogs: 1910 to 1949, 1949 to 1954, *Superintendent of Documents*, U. S. Government Printing Office, Washington 25, D. C., $2 and $1, respectively. Both contain directories of depository libraries in the U. S. at which USBM publications can be consulted.

• ORDER BOOKS SO MARKED THROUGH AIME— Address Irene K. Sharp, Book Dept. Ten percent discount given whenever possible.

NOW — Allis-Chalmers Vibrating Screens FLOAT ON AIR

NEW cars, modern trucks and trains utilize air springs for the smoothest, best controlled ride obtainable. Now Allis-Chalmers has applied the *same type air springs to vibrating screens* to provide greatly improved screen performance.

For complete information, see your nearby A-C representative, or write Allis-Chalmers, Industrial Equipment Division, Milwaukee 1, Wis. Ask for Bulletin 07B8931.

A-C ALLIS-CHALMERS

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A-5662

Manufacturers News

News
Equipment
Catalogs

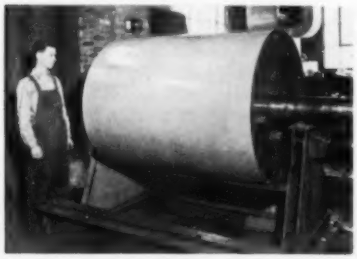
• FILL OUT THE CARD FOR MORE INFORMATION •

Air Spring Screens

Allis-Chalmers Mfg. Co. shows the way to reduce vibration and noise of vibrating screens with the development of new air springs constructed of nylon cord and tough rubber. Easily installed springs can be adjusted to weight differences by altering their air pressure. Inflate from main air line, pump, or air bomb. Springs are now available on new Allis-Chalmers floor-mounted screens in 5x12-ft or larger sizes. Can be installed in existing units if these are already floor-mounted with dual springs on each corner. **Circle No. 1.**

Magnetic Cobbing Pulleys

Uniformity of magnetic field is a result of special overhung poles in new pulleys for dry cobbing of ores and tramp iron removal by *Magnetic Engineering & Mfg. Co.* Unit shown is 48-in. diam, 64 in. wide, weighs 22,200 lb. Manufacturer recommends 42-in. diam for —6-in. +3-in. material, 18-in. diam for —⅜-in.+⅛-in. feed. **Circle No. 2.**



Heavy Duty Trucks

A new truck line by *Ford Motor Co.* features models with up to 20 pct greater payload capacity than previously offered in heavy Ford trucks. Included is a wide selection of tilt cab, conventional and tandem models ranging from 25,000 to 51,000 lb in weight. Durability is stressed. The new Ford haulers fill all requirements of over-the-road trucks. **Circle No. 3.**

Blast Hole Drill

Winter-Weiss Co. has a new Portadrill, Model 105TA, for blast hole and exploratory drilling in all strata. All power is supplied by engine of the

Eimco 105 diesel tractor which positions the unit. Patented shock-hammer break-out tongs speed making or breaking of drill stem joints. In sandstone and shale, the rig has a rated capacity of 300 to 400 ft of 9-in. holes per shift using standard roller cone bits. **Circle No. 4.**

Lightweight Drill

One-man underground exploratory work with quick set-up and tear-down is possible with the new Super Scout diamond core drill by *Diamond Drill Contracting Co.* Anchor column provides support at any angle. A 2-hp air motor supplies power. Total weight is 85 lb. **Circle No. 5.**

Tamping Poles

Austin Powder Co. has new lightweight tamping poles with rubber head blocks for faster loading of blast holes. Constructed of aluminum pipe, the poles are non-warping, non-splintering, and non-sparking. Lengths of 4 to 20 ft are available; hook and eye arrangement makes adding lengths easy. Rubber head blocks offered in 1⅝, 4½, or 5-in. diam sizes. **Circle No. 6.**

Sump Pump

The economical rubber-lined SRL-V vertical sump pump by *Denver Eqpt. Co.* is available in four sizes with capacities from 20 to 1400 gpm. Simple design and high efficiency means easy maintenance and low horsepower needs. **Circle No. 7.**

Continuous Weigher

A weighing device for moving dry materials is offered by *Stephens-Adamson Mfg. Co.* A circular trough balanced on a pivoting fulcrum, it employs a revolving turret equipped with blades to move material to an adjacent outlet point. Pressure is exerted on a mechanical linkage in proportion to volume fed. Continuous weight recording is made on an indicator head through the linkage. Weigher may be used with any conveying or feeding units. **Circle No. 8.**

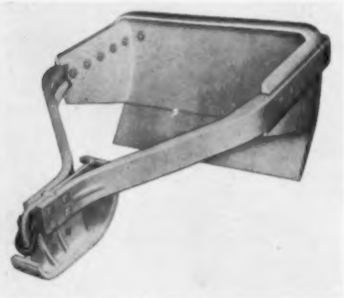
Low-Slung Mucker

All parts of a new mucker by *Wagner Tractor Inc.* are lower than the top of its tires, including the 1¼-yd capacity bucket, when in position for loading and carrying. Maneuverability in tunnels is aided by 4-wheel drive and 4-wheel steering, free oscillation of both axles, planetary drive, and hydraulic power steering. **Circle No. 9.**

(*Continued on following page*)

Scraper

Thompson-Berg Co. offers a new line of Superior scrapers in sizes 36 to 84 in. for use with slushers



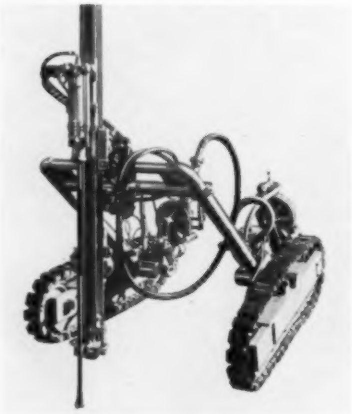
from 10 to 150 hp. Shoe is contoured and adjustable, slides easily and adapts to various digging angles. Extra-long side plates assure full capacity. Ribs on contoured back are drilled for multiple chain hook-up or single point hook-up for close breast digging. **Circle No. 10.**

Harder Cutting Edges

Caterpillar Tractor Co. has selected Hi-Electro hardening for best wearability and strength in scraper and bulldozer cutting edges. Process offers control needed to produce a superhard case and retain a tough shock-resistant core. Edge's projected area can also be kept thin for good penetration. **Circle No. 11.**

Air Drill

U-bar construction and large air motors outfit the LWD-400 Rock Cruiser by *Le Roi Div.*, Westinghouse Air Brake Co., for heavy drilling in rough terrain. Drill feed can be placed flat on ground, horizontal 7 ft above ground, or vertical. Each rubber-cleated track is powered by a 7½-hp air motor. Disc brakes automatically engage when machine is stopped, release on start-up. Available drills include 3⅛, 3½, or 4-in. Ten-ft feed travel allows 8-ft steel changes, optional feeds permit changes up to 18 ft. **Circle No. 12.**



New Pump Lining

Dorr-Oliver Inc. announces its Olivite and Oliver Diaphragm Slurry pumps are now lined with Hypalon synthetic rubber for greater resistance to abrasion and corrosion. **Circle No. 13.**

Round Drill Rods

A 1½-in. round hollow extension steel is made by *Brunner & Lay Inc.* especially for use on the crawler-type blast hole drill units. Carbon steel rods are furnished in 10 or 20-ft lengths, with K or 600 threads. **Circle No. 14.**

News & Notes

Roots-Connersville Blower, div. of Dresser Industries Inc., has completed a major expansion of its Connersville, Ind., main plant. . . . *Bemis Bro. Bag* Co. observes its 100th anniversary this year. . . *Caterpillar Tractor's* 400-plus dealer outlets will assure buyers they are getting quality used equipment by offering a guarantee backed by a $10,000 bond. . . . *Allis-Chalmers* plans new engineering, development, and research laboratories in Greendale, Wis., a Milwaukee suburb. . . . North American Cyanamid Ltd. has changed its name to *Cyanamid of Canada Ltd.* and will establish headquarters in Montreal.

RARE EARTH METALLURGY: A comprehensive survey of abstracts covering ten years of literature on ferrous and nonferrous applications of the rare earths has been incorporated in a 46-page publication of the *Davison Chemical Co. Div.* of W. R. Grace & Co. A complimentary copy of "Metallurgical Applications of the Rare Earths," may be obtained only on letterhead application to Davison Chemical Co., P. O. Box 488, Pompton Plains, N. J.

(21) HAULAGE LOCOMOTIVES: *Goodman Mfg. Co.* has a catalog, G-126, on two types of high-speed mainline underground haulage locomotives. Both the 27-ton type 201 and the 50-ton type 202 offer speed, power, and traction for long heavy hauls.

(22) COAL OPERATIONS: A 12-page catalog from *International Harvester Co.* shows how crawlers, scrapers, off-highway haulers, and engines can be used more profitably in coal field operations. Form CR-606-H, "Modern Methods for Coal Operations," supplies details.

(23) FLUID ENERGY GRINDING: Micronizer, a fluid energy mill for simultaneous grinding and classification to finest particle size, is detailed in bulletin 091 from *Sturtevant Mill Co.* Publication includes typical grinding data for various materials and unit sizes, pressure requirements, and capacities.

(24) VERTICAL SUMP PUMPS: New 4-page brochure 757 illustrates the Hydroseal vertical sump pump, latest member of the *Allen-Sherman-Hoff Pump Co.* line. Wearing parts are all metal or of rubber or synthetic material depending on intended use. Running clearances are protected by a small flow of gland water. Operating submerged, pump starts without priming. Three types: slurry, sand, and dredge. Three sizes: 1½ to 6-in., suction and discharge.

Free Literature

(25) WOOD TANKS: *National Wood Tank Institute* has made available a complete and up-to-date reference source on wood tanks. The 28-page indexed volume includes uses, capacities, properties and relation to expected service, and includes the use of wood to cover severe conditions by the use of selected polymer linings.

(26) TRACTORS: Versatility and economy is the theme of a new booklet on D7, D8, and D9 tractors

by *Caterpillar Tractor Co.* Eight-page form D764 illustrates with photos and job stories their wide range of applications.

(27) PHOTOGRAMMETRY: *Canadian Aero Service Ltd.* reports on the application of aerial surveys to engineering problems in a new brochure. Typical examples include photo analysis and reconnaissance mapping for highways, pre-engineering studies and staking for railroads.

(28) COMPANY CATALOG: The 34th Anniversary 1958 catalog of *Gardner Laboratory Inc.* has 102 pages of products of interest to laboratories and other testing units.

(29) MINE DRAINAGE WATER: Chemical methods for purging and maintaining mine drainage and coal preparation plant process water are covered in a report issued by *Saverite Engineering Co.* Saverite treatment has been extended to the water cooling systems used by many continuous miners.

(30) LUBRICANTS: Proper lubrication of open gears, dipper sticks, and cams is the subject of a new folder published by *Whitmore Mfg. Co.*

(31) OUTDOOR MOTORS: Features that give outdoor reliability to *Allis-Chalmers* weather-protected motors are described in bulletin 51B8606B.

(32) MAGNETIC SURVEY UNIT: A portable electromagnetic prospecting unit by *Kellogg Exploration Co.* features variable frequency operation. Operator can select present frequencies for transmitter to suit expected conductivity conditions. Battery-operated unit is self-contained.

(33) FLAT GRAB HOOKS: A new line of drop forged grab hooks which will "reduce chain kinking, fit the chain better, have less side movements and let the hook hang straight," is offered by *Crosby-Laughlin Div.*, American Hoist & Derrick Co., in catalog 950-2.

(34) HEAVY DUTY BRAKE: Built-in automatic adjustment assures constant efficiency throughout the life of a new brake lining by *B. F. Goodrich Aviation Products.* Hi-Torque is first hydraulic drum type brake with 360° expander tube actuation designed exclusively for big tractors, scrapers, earthmovers.

(35) METAL DETECTOR: The Twin Loop metal detector by *Stearns Magnetic Products* can be installed on any conveyor system handling non-magnetic materials. Inspection loops will detect tiny metal particles at speeds from 30 to 600 fpm. Details in bulletin 1101-X.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

4

Mining Engineering 29 West 39th St. New York 18, N. Y.

Not good after July 15, 1958—if mailed in U. S. or Canada

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

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51	52	53	54	55	56	57	58	59	60
61	62	63	64						

Students should write direct to manufacturer.

(36) BIG CRAWLER: *Allis-Chalmers* details major design advances of the HD-21 turbocharged diesel tractor in a new 16-page catalog, MS-1243. Longer track lends better balance, decelerator pedal makes for easier control, and matched torque converter and power train supply steady power under all operating conditions.

(37) FLEXIBLE COUPLINGS: Sure-Flex couplings by *T. B. Wood's Sons Co.* consist of two flanges and a two-piece rubber sleeve, which are combined without clamps or screws and are designed to withstand misalignment and end float. Five tables in a new 8-page bulletin allow shortcuts in coupling calculating and supply fast answers to application questions. Bulletin 10100 has the necessary information.

(38) TRACK PRESS: *Owatonna Tool Co.* claims crawler-track overhaul time can be cut in half by use of its new track press. The new press, OTC Trackmaster model Y7000, pushes pin and bushing simultaneously without broaching or damaging sidelinks. One man can take down and reassemble even badly rusted track. Unit can handle all makes and sizes of track with grousers on (two bolts removed) or off.

(39) POWER & CONVEYOR CHAINS: *Chain Belt Co.* offers its new catalog, 610, which covers a complete line of chain for almost all power transmission, timing, tension, and conveying applications. Basic selection data is included along with detailed information on each major classification.

(40) BLOWERS & GAS PUMPS: Increased ratings for the full line of type AF rotary positive blowers and type XA rotary positive gas pumps by *Roots-Connersville Blower* are supplied in bulletin AF-XA-157. Design refinements permit higher speeds, greater efficiency.

(41) COMPANY STORY: *Metal Carbides Corp.* gives readers a pictorial tour of its Youngstown, Ohio, plant in a colorful 24-page folder. The tungsten carbide plant story begins with the reduction operation and proceeds through raw material preparation, pressing, storage, analysis, heating, pre-forming, weighing, and machining.

(42) CYCLONE: Brochure 1157 from *Heyl & Patterson Inc.* supplies information on the H & P cyclone, and includes tables, graphs, exploded views, and photos of field installations.

New Films

Bucyrus-Erie Co. shows the operation of its big all-hydraulic truck-mounted model H-5 Hydrocrane in *Hydro-Magic*, a 16-mm sound color motion picture, No. S-406, which runs 23 min. The Hydrocrane's precision control job-simplifying features and attachment uses are demonstrated in on-the-job scenes. Also shown is the Hydro-railer, a Hydrocrane fitted with steel railway wheels which set hydraulically. The film is available upon request, given on company letterhead, from *Bucyrus-Erie Co., Publicity Dept., South Milwaukee, Wisc.*

Memo To The Winning Contractor . . Why Twins? **is a new 16-mm color and sound movie from Euclid Div., General Motors Corp. Showing operations of Euclid's twin-power 24-yd struck capacity TS-24 scraper, the film points up the decided trend to twin-powered units for many kinds of earthmoving jobs and spells out where, how, and when twin power can be put to work advantageously. The film which is available for showing to interested groups and organizations may be obtained by writing the *Euclid Div., General Motors Corp., Cleveland 17, Ohio*, or through Euclid regional offices.**

Battelle Memorial Institute Cobalt Information Center announces the availability of free 16-mm sound films to technical, research, industrial, and educational groups. *A Study of Cobalt Deficiency in Ireland*, a 27-min 16-mm sound color film made in Ireland by the Mond Nickel Co., depicts the need for cobalt as a trace element in the diet of ruminant animals—in this case, sheep. Filmed by the British Oxygen Co. Ltd., *Depositing Stellite with Oxyacetylene Flame*, a 20-min 16-mm sound film, is a technical instruction movie presenting the method of hard facing with Stellite. Three grades of Stellite are shown, explained, and their uses depicted. *Cobalt in Katanga*, a 35-min 16-mm sound color movie, presents a step-by-step description of cobalt production in the Belgian Congo. Operations in the Congo—at Musonoi Mine, Jadotville-Shituru Electrolytic Plant, and Panda Electric Smelter Works—and the Olen plant in Belgium are shown. The film traces production methods from the blasting of the Co-Cu ore through mining, concentration, leaching, reduction, and refining to the metal: granules and rondelles. The three films are available for viewing in the United States and Canada. For reservations write, on company letterhead: *Cobalt Information Center, Battelle Memorial Institute, 505 King Ave., Columbus 1, Ohio.* It is suggested that several suitable dates be outlined in the letter.

To show how finish grading can be done faster, easier, and with greater economy, Caterpillar Tractor Co. has produced *The Preco Automatic Blade Control*, a 5-min 16-mm film. Filmed under actual job conditions, the picture shows a versatile attachment for No. 12 and No. 112 Motor Graders. Showing of the film can be arranged through local Caterpillar Dealers or by writing the *Advertising Div., Caterpillar Tractor Co., Peoria, Ill.*

Energetically Yours, **a new color cartoon 13-min 16-mm film, available for free loan, has been produced by Transfilm for the Standard Oil Co. (New Jersey). The animated film traces the search of man, from primitive to modern times, for new sources of energy and how each discovery has altered man's way of life. Designed by the British cartoonist-satirist Ronald Searle, author and *Punch* cartoonist, the film traces man's "energy" progress from his discovery of animal power through the atom's release of energy. *Energetically Yours* was premiered late in 1957 on Jersey Standard's 75th anniversary network television program. Requests from business and community organizations or schools for a free-loan showing of the film should be directed to *Standard Oil Co. (New Jersey), Room 1610, 30 Rockefeller Plaza, New York 20, N. Y.***

Merge and Diversify?

American Potash & Chemical Corp., a strong firm in the lithium, rubidium, and cesium fields, and Lindsay Chemical Co., a rare earths and thorium chemicals heavyweight have agreed to merge, directors report. Shareholders to review plan April 29. . . . Nine companies in the metals and precision manufacturing field have been consolidated in Engelhard Industries Inc. Move, which includes a top producer of platinum, Baker & Co., puts Engelhard into the 200-biggest category of U. S. companies. . . . Haile Mines Inc., tungsten and manganese ore producer, and Howe Sound Co., lead-zinc producer and biggest U. S. cobalt miner, have jointly announced merger plans. Haile earlier this year acquired Frank Samuel & Co., ore mineral and chemical importers, Refractories Corp. of America, National Paint & Manganese Co. and, in February, controlling interest in Karl Lieberknecht Inc., manufacturer of textile machinery. Move, say spokesmen, will put consolidated firm in strong position to further integrate and diversify present operations.

Metal Prices Change, Most Falter

Platinum—Baker & Co. has chopped its price for platinum by $5 an ounce to the $72 to $75 range. Change reflects effects of recent Russian metal offerings. **Antimony**—In the first price change since a 4½¢ raise in August 1955, antimony has been cut 4¢ a pound, by National Lead Co., to 29¢ for bulk quantities. Poor demand was blamed. **Selenium**—Common grade selenium in bulk lots has been reduced from $7.50 to $7 a pound by American Smelting & Refining Co. High purity metal is down $1 to $9.50. **Mercury**—The only recent upturn was one for mercury, which advanced about $5 to $226 a flask for large lots. Dealers sold small quantities for as much as $230 a flask. Industry lays strength to Government purchases, London market firmness.

Du Pont Takes Over Ilmenite Mines

In a move toward more closely-knit company operations, Du Pont Co. has taken control of its two Florida ilmenite mines from Humphreys Gold Corp., which started up the mines for Du Pont 10 years ago. Mines—the Trail Ridge and Highland—produce the black sand for use as white titanium dioxide pigment and raw material for titanium metal.

Ungava Nickel Exploration Dropped

American Smelting & Refining Co. reports its nickel-copper exploration of the Ungava Peninsula in Quebec "has failed to indicate sufficient potential tonnage and grade to justify an integrated nickel operation." Asarco will not conduct further exploration in 1958 but has not decided on final disposition of the concession.

Coal's 1957 Markets

A director of the National Coal Assn. summarzied coal's 490 million ton market in 1957 as follows (millions of tons): Electric utilities, 157.4; Steel, 112.9; Other industrials, 89.9; Retail, 36.2; Cement, 8.8; Railroads, 8.4; Canada, 19.3; Other exports, 57.0.

(Continued on following page)

Uranium Complaints Under Scrutiny

Uranium miners, hard hit by the AEC's October 1957 announcement that uranium output would be limited to mill contracts under negotiation, have come up with several problem-easing proposals: 1) An open market for the metal, with privilege to export; 2) A stockpile program to ease producers up to the time, presumably 1962, when industrial demands will take up the surplus; 3) No extension of foreign contract commitments. (Ed. Note: Average U. S. price this year has been estimated at \$9.60 per pound of concentrate, import price at \$11.15 per pound.) AEC is surveying industry conditions, has already expressed opinion that stockpiling would "weaken" the industry.

Fastest Shaft Sinking Yet

A 262-man crew working three shifts recently sank a South African gold mine shaft from 199 ft to 1033 ft level—834 ft—in a 30-day period. In addition, the shaft—No. 2 of the Free State Saaiplaas Gold Mining Co.—was lined with concrete for 810 ft. Largest South African shaft, the Saaiplaas is 29 ft 6 in. diam. The new world record exceeds the old mark, held by the Monarch shaft of West Rand Consolidated Mines, by 71 ft.

New Process for Electrorefining Nickel

International Nickel Co. of Canada has developed a new electrorefining process which includes the direct electrolysis of nickel matte. Eliminating high-temperature oxidation and reduction operations, the process involves taking low-copper nickel sulfide from Bessemer converter or other source, casting directly into sulfide anodes, and electrolyzing. Process permits commercial byproduct recovery of elemental sulfur and selenium.

Cerro Looks to Lower Output

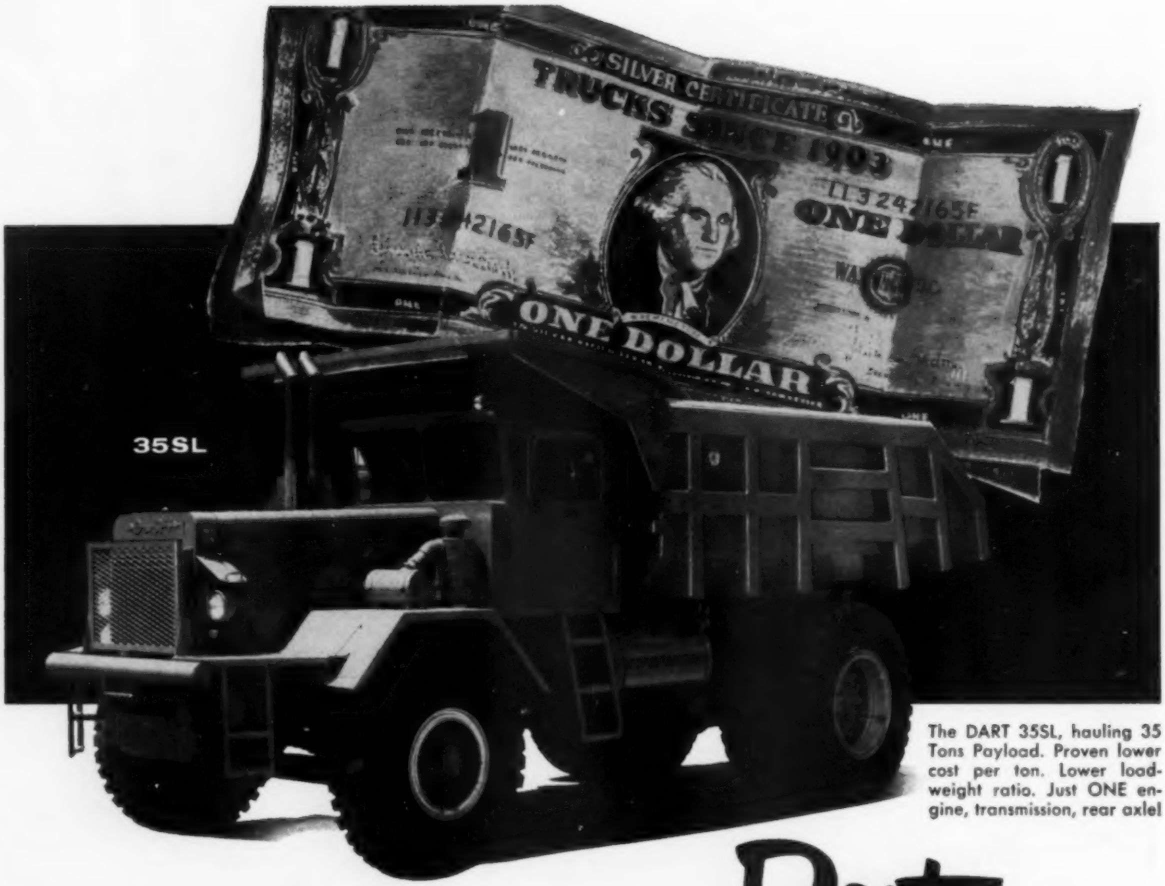
Estimated 1958 output of copper, lead, and zinc at the Oroya, Peru, smelter and refinery will be some 7½ pct under last year, reports Cerro de Pasco Corp. The cutback adds substantially to the 300 ton per month slash announced by Cerro last December.

Another Big Kennecott Cutback

A production- week cut from 6 to 5 days will reduce the output of Kennecott Copper Corp.'s four Western Mining Divisions by 12½ pct. Added to previous reductions, the curtailment results in a total slash of 20½ pct from former capacity operations.

Further Base Metal Output Cuts

St. Joseph Lead Co., with the aim of cutting March production of lead by 6000 tons, offered its Bonne Terre, Mo., workers the following alternatives: a three-week vacation shutdown, a four-day week for 15 weeks, or shutdown of high-cost operations. Worker choice was three-week closing. Operations at the company's Balmat and Edwards zinc mines in New York were also slowed. . . . American Metal Climax, because of the price slump has deferred full operation of its 75-pct owned subsidiary, Heath Steele Mines Ltd. The lead-zinc-copper mine and plant, with rated capacity of 45,000 tons of ore per month, will continue breaking-in operations at less than one-third of capacity.

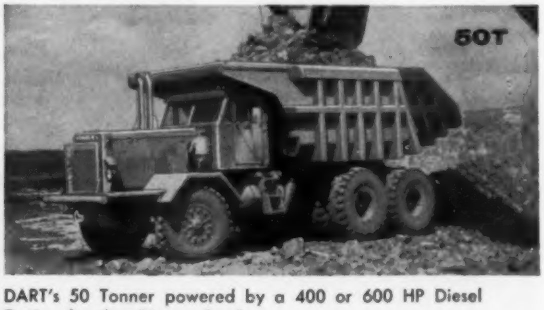


The DART 35SL, hauling 35 Tons Payload. Proven lower cost per ton. Lower load-weight ratio. Just ONE engine, transmission, rear axle!

PROFITABLE PAYLOADS WITH Dart TRUCKS

DART TRUCK COMPANY builds a complete line of trucks—from 10 to 50 Ton payloads. *But*, any model can be built to meet your special haulage requirements.

DART 13S-UG underground shuttle truck with dual controls and through conveyor.



DART's 50 Tonner powered by a 400 or 600 HP Diesel Engine for that big payload.

WRITE FOR SPECIFICATIONS TODAY

DART TONNAGE ENGINEERED TRUCKS
Kansas City 8, Missouri

Bulletin! March 14 Dart merged with Kenworth Motor Truck Division of Pacific Car and Foundry Company—new name: KW-Dart Truck Company.

D-140

...More about 2 proved ways you can Modernize to Economize

The 40-R Rotary DOES Cut Drilling Costs — in quarry after quarry, with features like these: air bailing cools the bit and cleans out cuttings. Remote-controlled, power-driven drill pipe rack reduces manual effort. Hydraulically powered down pressure on the bit provides controlled load on drilling tools for maximum penetration. Ward Leonard control on rotation of the drill pipe permits drilling at the most efficient speed for a given formation. No wonder those who use them want to convert completely to Bucyrus-Erie rotaries!

Let Us Estimate the Savings to You — Trained Bucyrus-Erie men will study your operations and show you how you can use the 40-R to effect real cost savings — *even with relatively low production requirements.* You have nothing to lose — and much to gain — by letting us *prove* what the 40-R will save you.

The 110-B Ward Leonard Electric Shovel has been known as a low-cost performer wherever it is used — particularly on jobs involving big yardages, or extra tough materials or conditions. It is not a "stretched" machine. It was designed from the ground up to handle 4½ yards in tough going. The outstanding performance of the one shown here loading trap rock in New Jersey made the owner a repeat buyer.

Give Us an Opportunity to Explain Its Savings. We would like to show you production and cost figures which tell best how the 110-B performs and how it saves money.

A Bucyrus-Erie 110-B 4½-yd. electric shovel loads trap rock.

This Bucyrus-Erie 40-R rotary drills 9-in. hole in dolomite.

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BUCYRUS ERIE

A Familiar Sign at Scenes of Progress

BUCYRUS-ERIE COMPANY
South Milwaukee, Wisconsin

Mine Blasting With A Countdown?

Problem: Crush 400,000 tons of rock underground with a single blast.

Impossible? The Atomic Energy Commission using one of the baby bombs in its brood of nuclear weapons did exactly that one day last fall. And the deep underground detonation, labeled *Rainier*, has been given a close look by industry on the chance that the shot will—perhaps—supply data for practical application in the future.

Rainier required two major elements. First, the bomb, a device with a known yield of 1.7 kilotons (the equivalent of 1700 tons of TNT); second, the blast site, a tunneled volcanic rock mesa on the northern edge of Yucca Basin at the Nevada test site.

The AEC undertook the project after considering the attractive advantages of an underground shot: fall-out would be nil, blast and flash effects would be shielded by the bomb's mountain container, and the radioactive clouds that play hob with air traffic schedules would be nonexistent.

After much involved calculation, the Commission decided on an economical horizontal tunnel that would pierce 1900 ft of the mesa's *Oak Springs tuff* interior. Tunneling crews drilled a burn footage of 69,300 ft and mucked out with a front end loader. Near the tunnel end a right angle section was driven. A short distance farther another right angle drive was made. The adit was then advanced in a curved line until a point midway in the *hook* thus formed was reached. Hook-shaping the tunnel end provided for self-sealing of the tunnel to prevent any escape of blast effect. Into this final section, in a chamber about 900 ft under the mesa top and some 800 ft from its steep sides, the small nuclear device was placed and subsequently, on September 19, it was detonated.

Observers at a 10-mile distance noted dust spurts as dislodged rocks rolled down the mesa slopes and some saw the mountainside ripple as the shockwave moved upward. Certain observers felt a slight earth tremor, others felt nothing—although seismological stations recorded earth waves as far away as College, Alaska—2320 miles.

Later, core drilling from the top of the test mesa showed that a large mass of broken rock was centered about ground zero. In addition, it was deduced that blast force and heat had briefly formed a 110-ft diam *glass bubble* at the center of detonation. The weight of shattered rock above this cavity, however, had quickly crushed the sphere and almost filled the space it had occupied.

Core drilling was stopped when the ground zero level was reached and horizontal drilling was begun from the still open end of the tunnel. Three and a half months after the blast, drillers encountered rock temperature of 113°F at 60 ft from ground zero, some 50° above normal. Readings from a distance of 35 ft to the point of detonation indicated a fairly constant temperature of 90°F. Highest temperatures —up to 190°F—and maximum radiation—45 roentgens per hr—were recorded below the blast center.

There had been some theorizing before the blast that the great energy released would cause the formation of precious stones of some type. But so far the AEC has not turned jewel merchant and it has found no evidence that any gems were created.

As part of *Project Plowshare*—an investigation into the possible application of nuclear detonations for peacetime purposes—the Rainier blast provides some interesting ground for speculation. The AEC view is reported here:

> It has been calculated that the underground shot produced at least 50,000 tons of permeable broken rock and an additional 400,000 tons of crushed, but relatively impermeable rock. The large masses of broken rock suggest such applications as use in mining to break up orebodies for removal or leaching, and in oil strata to free crude oil trapped in relatively nonporous rock formations. It also is considered possible that heat resulting from an underground detonation in oil strata might increase the production of oil in certain situations by making it flow more freely through the rock formation.

The Commission believes such projects could conceivably have other peacetime applications.

1) Water could be piped into a rock formation heated by a contained nuclear detonation to form steam for power.

2) To effect composition changes, various materials could be placed around a device to be detonated deep underground.

3) Makeup of the earth's crust could be seismically studied through bomb-initiated earth waves.

4) Earth could be moved in great quantity, as in excavating for a large canal.

Interesting though the project may be, the possibility of atomic blasting in mining evokes many questions. Although improvements can undoubtedly be made, this first test case gave results that were high in quantity but low in quality. Moreover, the tufa composing the test mesa (and the AEC points this out) is a very porous formation—not at all typical of the materials usually encountered. Then how would typically tough strata react to such blasting? How expensive would the blasting be? Furthermore, how long would the broken rock and its overburden be radioactive? As for mucking out, who would do it and how?

Use of the method for mining alone was not, of course, the purpose of the Rainier shot and the Commission is not being asked for answers. But, if another test is arranged for, perhaps it can be worked out in a way that will provide data applicable to more typical conditions.

Ten Years of Coal Progress

The Voice of America a short time ago asked the executive vice president of the National Coal Assn., T. Pickett, to supply a brief statement on recent advances in mechanizing the bituminous coal industry. Mr. Pickett complied with a summary that makes a number of interesting points. He notes that the average American bituminous miner turned out more than 10 tpd in 1957, as compared with a figure of 2½ tpd in 1890—and with the 2 tpd now produced by miners in the Soviet Union. And this increase per

man-day has been more rapid than in any comparable heavy industry.

Also, he notes, the wage rate has risen from $1.64 an hr to $3.06 an hr since 1947—again the result of increased mechanization. Safety has likewise improved. In the 1947 to 1957 period fatal injuries were reduced more than 50 pct and non-fatal injuries by more than 65 pct.

What is the percentage of mechanization? About 95 pct underground, says Mr. Pickett, 100 pct in strip operations.

Plant Spending on Downturn

If you are a member of the executive finance committee of an average mining company, the chances are that you and your colleagues have hedged your company's plans for plant and equipment spending by 15 pct from last year's total. And you are not alone in planning to ease up on expenditures. Businessmen the country over have reduced their capital outlay schedules—to $32 billion from the record $37 billion they spent in 1957, a reduction of 13 pct.

With businesses disposed to wait for changes in consumer mood, the bloom is off the boom in capital investment. A recent Government survey reports all major industries, with the exception of public utilities, expect to reduce their outlays in 1958 as compared to 1957.

In 1954 only $27 billion was set forth for new plants and equipment but this total climbed almost 40 pct by last year. Now the upward move is at least temporarily hindered. A reduction of one-sixth is expected by manufacturing firms, while commercial organizations look to one-eighth less spending. Railroads anticipate going down by more than a third.

No Secrets

The most interesting annual report received this year was not intended for us at all. Koehring Co. has taken a step which is not unique but is the first sample we have seen of an annual report honestly and realistically for employes. Its detailed description and breakdown cover the company's divisions, what they manufacture, and where the products go.

Following a letter from the president, a full report compares the safety records of Koehring's various divisions. This is followed by a graphic and human presentation of how operating costs arise, with an explanation of each item such as "cost of tools wearing out," "cost of taxes," "cost of human energy." Two pages lead off with the proposition that if "You own a business equal to one employee's portion of Koehring Company, it would be worth. . . ." This outlines what the individual would have spent, what he would have received, and what he would have had as profit at the end of the year. Koehring even explains the workings of LIFO accounting.

We read this report with interest and profit. We imagine that people with a daily bread and butter stake in that company read it even more closely and got even more from it. Hats off to Koehring.

Tariffs, the Loose End

Consensus means agreement of opinion and consensus is something the mining industry seems to have on only one point: Mineral and metal prices are too low for profitable and successful operation. There is close agreement on some things to do about it—on taxes, labor policy, and a very few other areas. But when you get down to tariffs there is wide disparity of company opinion, largely reflecting differing degrees of dependence on foreign investment.

Reviewing a series of clippings taken from a several-day assortment of New York newspapers, one gains an impression of even greater diversity of ideas than actually exists. This is what the public hears.

Some spokesmen favor high tariffs, protective tariffs; some are neutral or at least say as little as possible; others emphasize the value of imports and stress that tariffs must remain low and free trade must be encouraged. Without attempting to resolve the question, without attempting to assign the label "majority" to any one opinion, we would rather have the public see a uniform sampling of views than a sampling in proportion to the number of speakers on each side.

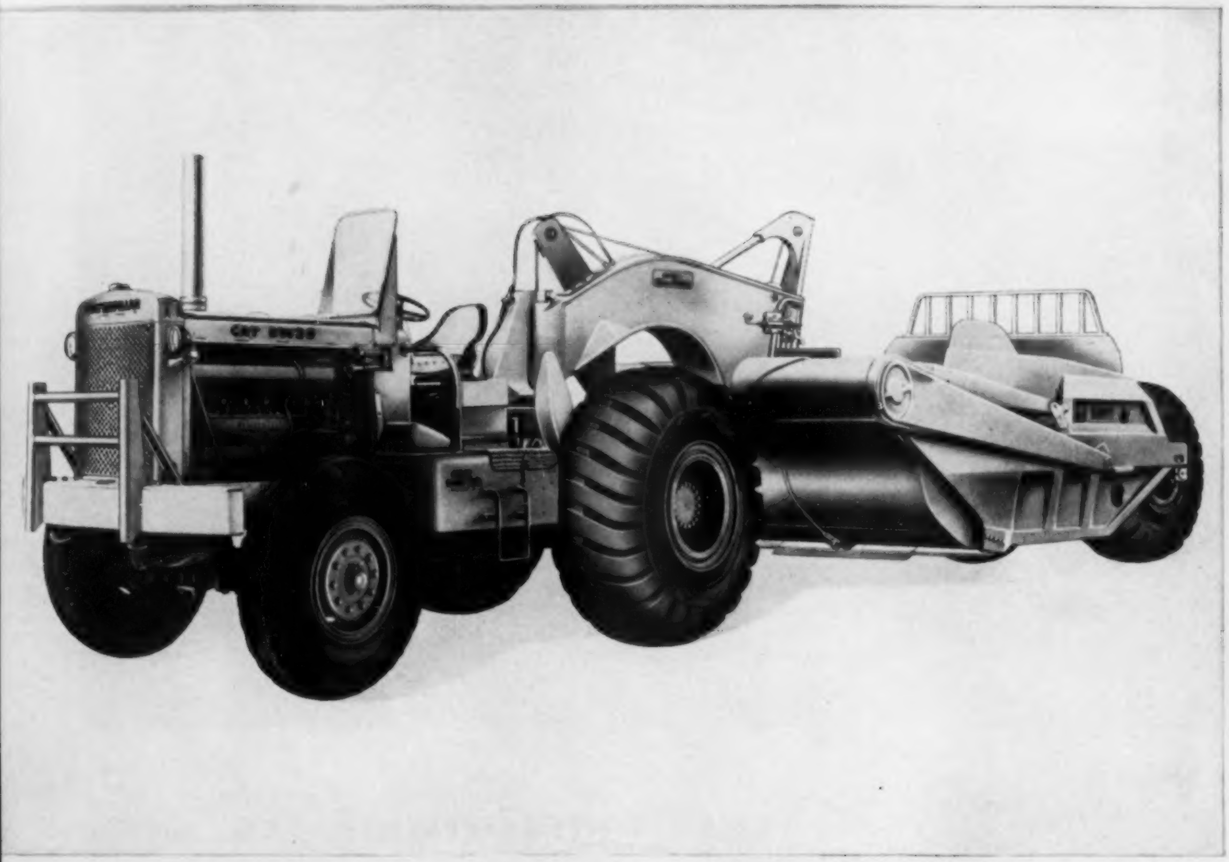
An example of this scattering of comment may be found in a single issue of one newspaper. A front page story carried a strong endorsement of the administration's proposal for a five year extension of the trade agreements act as brought out in testimony before the House Ways and Means Committee by R. P. Koenig, president of Cerro de Pasco Corp. He stressed the impact of U.S. tariff decisions on foreign economies, the importance to the U.S. of income derived from metal exports, and significance of imports of raw materials in which the U.S. is not self-sufficient.

On another page of this same issue, Andrew Fletcher, president of St. Joseph Lead Co., was quoted as saying "St. Joseph Lead Co. believes that it is to the benefit of the U.S. to stem the unparalleled flood of unneeded imports which has brought stress to mining communities and threatens to impair the development of our future latent mineral resources." Fletcher hoped that the Tariff Commission and Congress would "take the necessary constructive steps to protect the domestic mining industry from excessive imports."

Mr. Koenig of Cerro de Pasco recommended revising the escape clause language in the trade agreements act to make a distinction between ". . . questions arising from importation of raw materials of which the U.S. cannot possibly be self-sufficient, as opposed to questions arising from other items of which this may not be said." While not completely on opposite sides of the fence, these two statements still leave a nice question as to what defines self-sufficiency—and at what price.

On still another page this same newspaper reported formation in Colorado of an association to "fight for changes in Federal laws . . . to save the now dying small metal mines industry. The new organization, American Metal Mine Owners Inc. listed among its trustees Dan Thornton, former Governor of Colorado. J. F. Little, chairman of the Governors Committee on Natural Resources and another member of the new association cited its aims: "Since the miners' problems are chiefly political, this bi-partisan group should be of great help in opposing the policies of the Federal Government which have just about wrecked metal mining." While no mention of tariffs is made, the story indicates that the new organization will work with any other group to remove federal restrictions on mining, to end government price fixing on metal production, and to halt federal encouragement of foreign competition with domestic miners.

HERE THEY ARE!
DW20 and DW21 WITH



DW20
(SERIES F)

New HP—320 (maximum output)!
New Torque—28% torque rise!
New Speeds—up to 35.8 MPH!

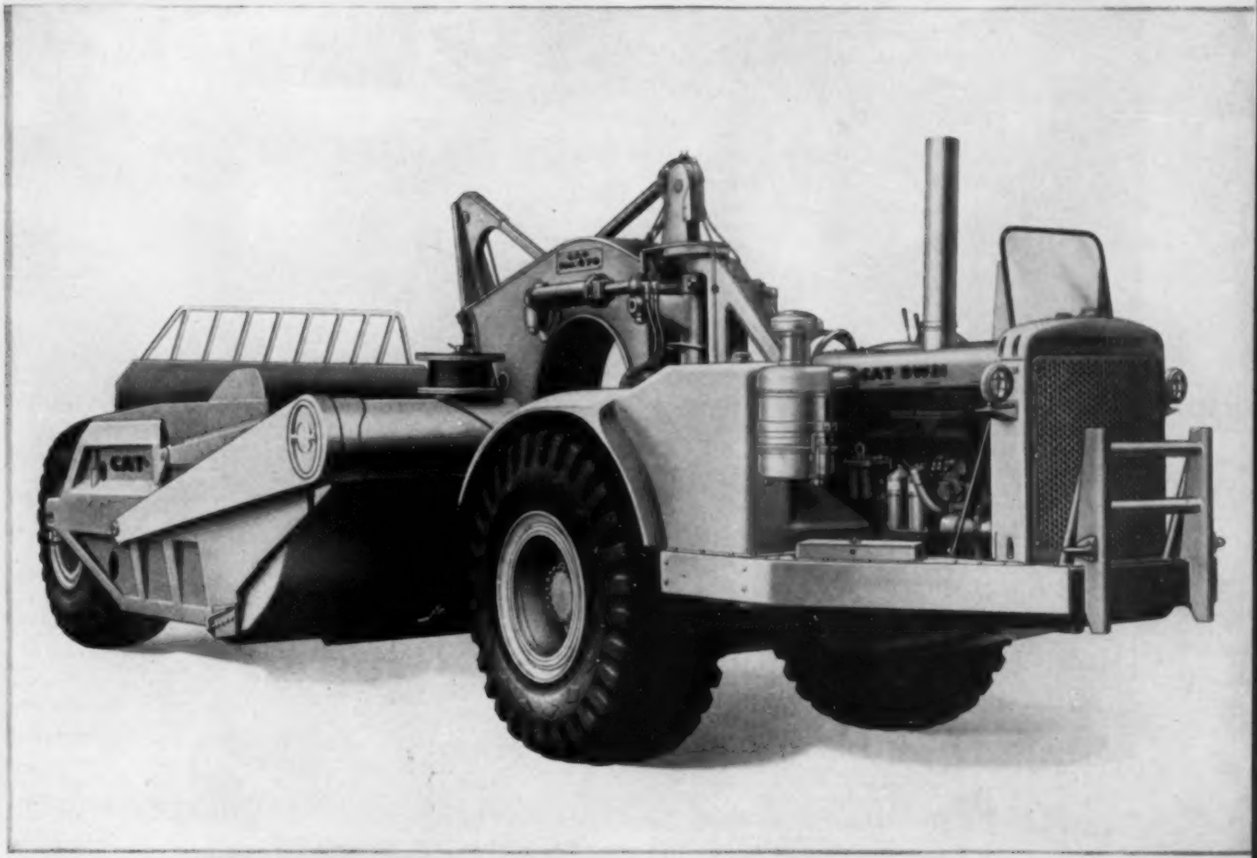
Powered with a brand-new SUPER-TURBO Engine, these big Caterpillar rigs are faster, more powerful than ever. Horsepower of both the DW20 and DW21 is increased to 320. At equal rimpulls, speeds are 10% greater, and torque rise is more than double compared to the previous models. Now you can move earth faster and easier—and still have the dependability for which Cat equipment is famous.

Behind the new performance of the new DW20 (Series F) and DW21 (Series D) is their new SUPER-TURBO Engine. This amazing power plant incorporates a new concept in diesel engine turbocharging. Its heart is a revolutionary air induction system, unique in earthmoving machines . . . and another Caterpillar first.

This system allows use of more of the Turbocharger's potential than was possible before. Results: twice as much torque rise, higher horsepower, better acceleration and gradeability.

But more important, these results translate into faster cycles, greater production and more profit—*for you.*

NEW INCREASED HORSEPOWER



DW21
(SERIES D)

New HP—320 (maximum output)!
New Torque—28% torque rise!
New Speeds—up to 22.6 MPH!

Add to these DW20 and DW21 power advantages all the design features of Caterpillar LOWBOWL Scrapers. These matched units are built to use the greatest possible loading potential of the new power in the tractors.

Higher horsepower! New torque rise! Faster working speeds! Your production will hit new highs and your costs will drop when you turn these improved rigs loose on your job.

Call your Caterpillar Dealer right now and set up a demonstration on a job of your choice. The sooner you see the powerful new DW20 and DW21 in action, the sooner you'll start getting bigger returns from *your* wheel spread.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

CATERPILLAR

SUPER-TURBO ENGINE
—First in the Industry
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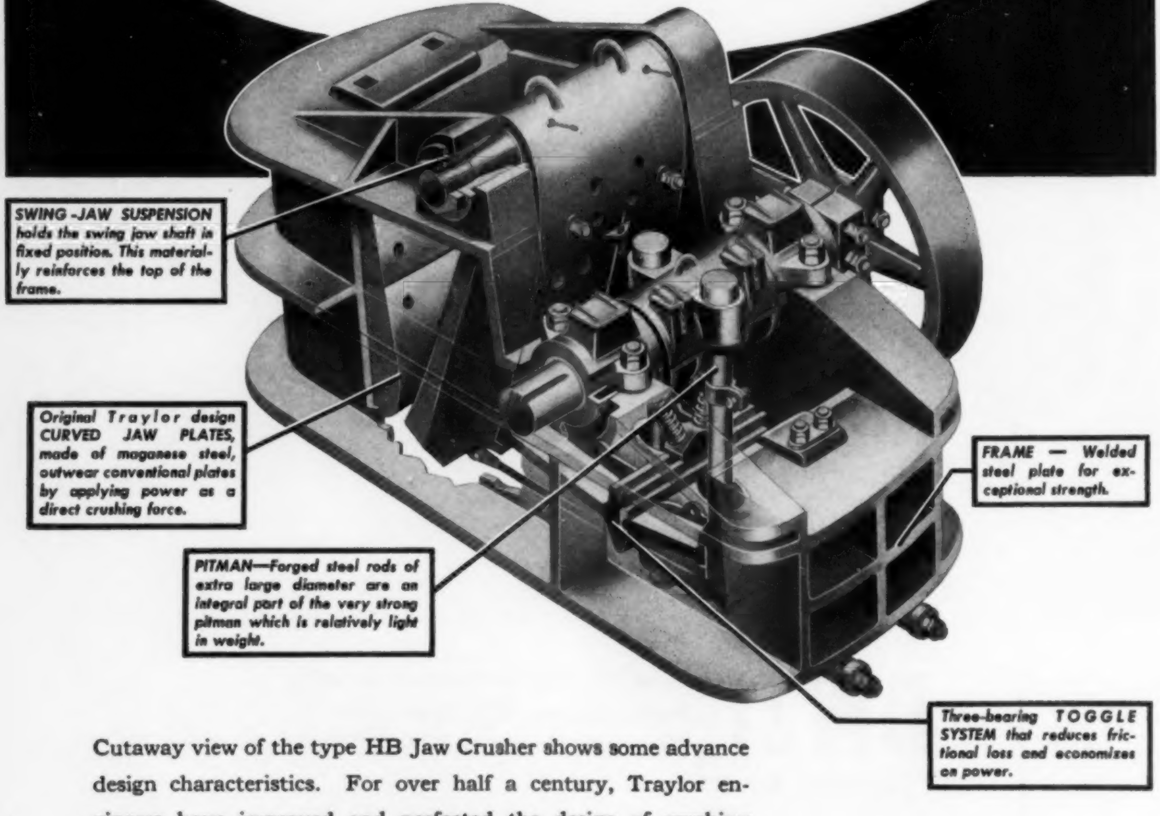
Traylor-MADE HB JAW CRUSHER



Cutaway view of the type HB Jaw Crusher shows some advance design characteristics. For over half a century, Traylor engineers have improved and perfected the design of crushing machinery. Call on Traylor engineers to help you solve your crushing problems. Write for bulletin No. 5105 today!

Traylor

TRAYLOR ENGINEERING & MFG. CO., 1033 MILL ST., ALLENTOWN, PA.

Sales Offices: New York — Chicago — San Francisco

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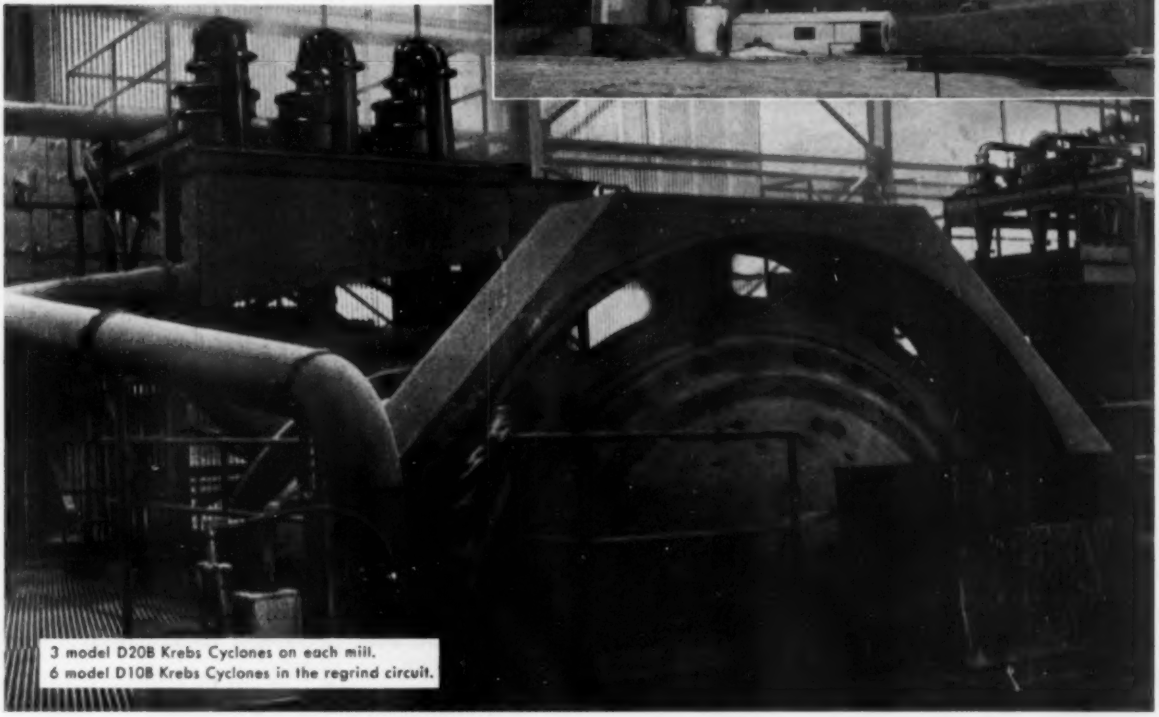
New copper concentrator does all its classification with cyclones



3 model D20B Krebs Cyclones on each mill.
6 model D10B Krebs Cyclones in the regrind circuit.

Pima Mining Company's Arizona plant, starting up in late 1956, is the first U. S. copper concentrator to classify entirely with cyclones.

Pima and nine other U. S. and foreign copper concentrators now have Krebs Cyclones for all or a major part of their tonnage. Operational costs can be substantially lower than with conventional classifiers. Capital cost is about 30%. The metallurgical advantages are usually the primary consideration . . . the flotation engineer may now have an optimum pulp density, and a selective grind of middling grains that reflects in increased concentrate grade as well as lower tailings.

Krebs Cyclone Bulletin 830 describing further cyclone techniques is available on request.

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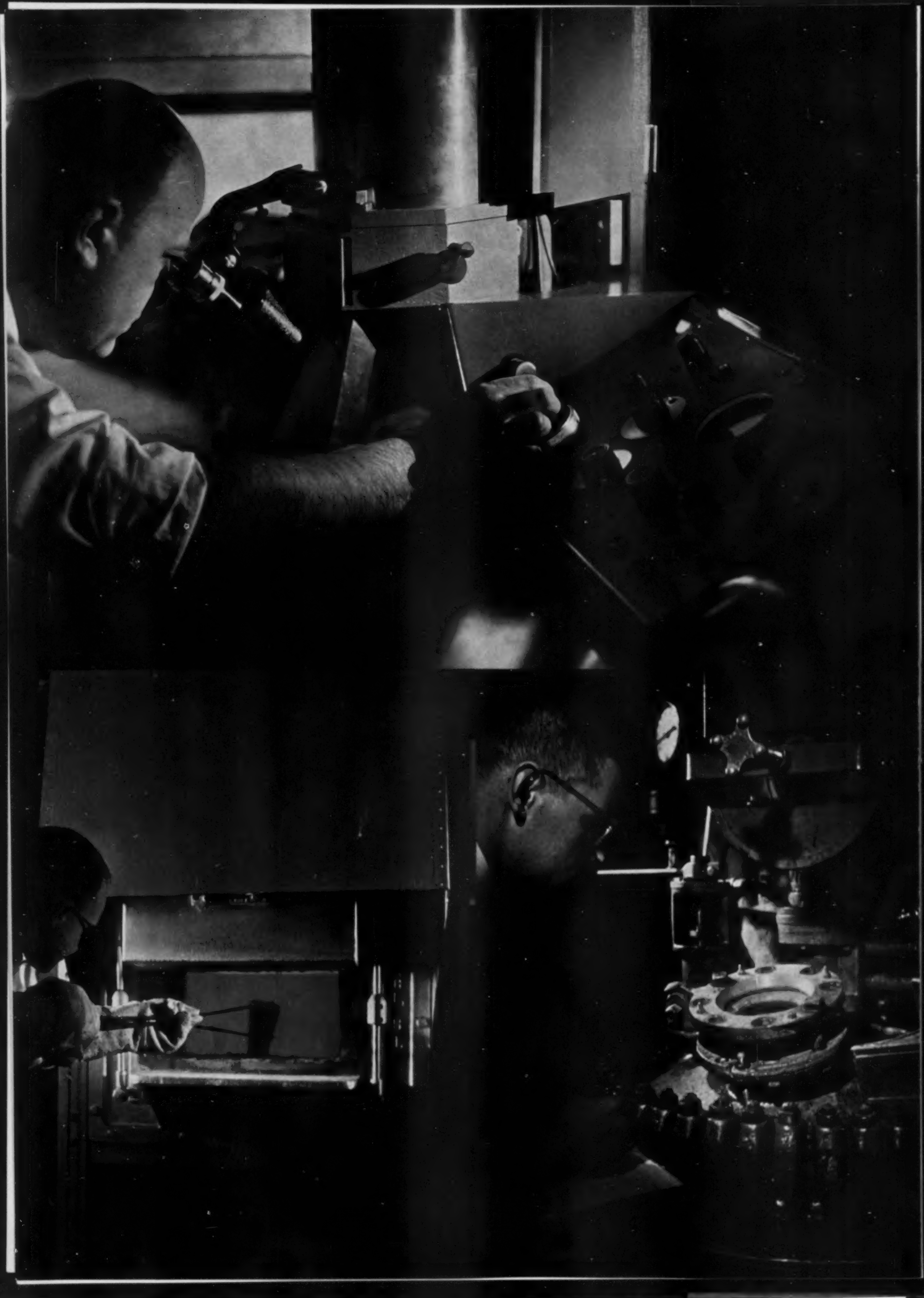
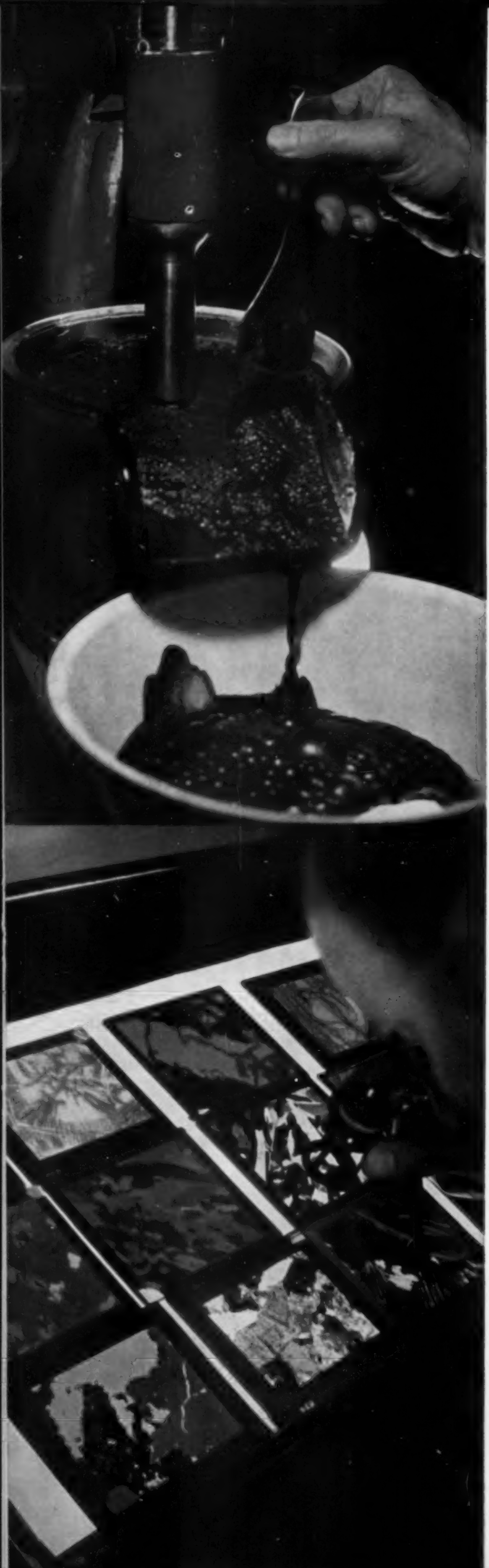
EQUIPMENT ENGINEERS INC.

41 SUTTER STREET SAN FRANCISCO 4, CALIFORNIA

Manufacturers of Krebs Cyclones, Valves and Clarkson Feeders





MINING engineering

DRIFT

Progress Report— United Engineering Center

As the picture on page 450 demonstrates, progress continues to be made toward completion of the new United Engineering Center to be located in New York on United Nations Plaza between 47th and 48th Sts. Razing of the structures on the site was completed in late March and actual groundbreaking for construction of the building is expected to take place later this year. MINING ENGINEERING's cover artist Herb McClure chose to compare the present Center on 39th St. with a silhouette of the architect's rendition of the future Center to highlight the progress to date.

As was reported in the February issue, page 179, the fund drive for the Center is well underway. The honorary chairman of the overall fund drive is the Hon. Herbert Hoover, 31st President of the United States and AIME Senior Past-President. In a speech at the opening dinner of the campaign in November, he emphasized the continuing—and increasing—need for scientifically trained personnel for this Nation's future. The role of the professional and technical societies and organizations in this future cannot be overstressed. Their role in coordinating and disseminating information is a vital one and the new Center will afford ample scope for their future growth and operations.

The fund campaign is basically divided into three parts: Industry, Member, and Greater New York City Commerce. Honorary vice chairman of the Business-Industry campaign is Alfred P. Sloan, Jr., former chairman of the board of General Motors Corp. Dr. Mervin J. Kelly, Bell Telephone Laboratories Inc., is directing this phase of the fund campaign, the first stage of which is well in progress. Working with him are AIME members H. DeWitt Smith, Leo F. Reinartz, Carl E. Reistle, Jr., Donald H. McLaughlin, W. B. Stephenson, and Will Mitchell, Jr.

The Societies, Member Gifts Campaign has, as honorary chairman, Charles F. Kettering. Joseph L. Gillson, E. I. du-Pont de Nemours & Co. is overall chairman of AIME's part of this phase of the fund drive. Working with him are Philip Wilson, for the Society of Mining Engineers; O. B. J. Fraser, for The Metallurgical Society; and M. L. Haider, for the Society of Petroleum Engineers.

The Greater New York City Commerce campaign is under the chairmanship of William H. Byrne. The construction of the new Center is to be financed from the proceeds of the

(Continued on page 450)

Drift

(*Continued from page 449*)

various phases of the fund drive: $5 million from industry-business and $3 million from the Societies' membership gift subscriptions. There is an additional estimated $2 million available through the assets of United Engineering Trustees Inc. It is expected that the goals of the fund drive will be reached in about two years and that the new Center will be ready for occupancy in the fall of 1960.

Col. Clarence E. Davies, retired secretary of ASME, is acting as coordinator for United Engineering Center plans. Architects are Shreve, Lamb and Harmon Assoc.; engineers are Jaros, Baum and Bolles for the planning stage; and Seelye, Stevenson, Value and Knecht are engineers for the structural stage.

The Center itself, to be located on United Nations Plaza between 47th and 48th Sts. in New York, will overlook the UN headquarters and East River beyond. Nearest neighbor to the south is the already operating Carnegie Foundation headquarters. Present plans call for a 20-story tower surrounded by a lower structure with landscaping appropriate to what now exists in the area. The new Center will be so constructed that it will not only have adequate facilities for the present functions of the Societies but will also provide for continuing expansion to solve growth problems of the engineering profession.

Present owners, through UET, of the 39th St. Engineering Center are the four Founder Societies—AIME, ASME, ASCE, and AIEE—and AIChE. Land for the new Center was acquired in the summer of 1957, after rebuilding on the present 39th-40th St. site proved impracticable.

Demolition proceedings at the site of the new United Engineering Center are witnessed by Walter J. Barrett, UET president, and William H. Byrne, general chairman of the Greater New York City Commerce Campaign for funds for the Center. Completion Goal: 1960.

"Not merely to sell; but to serve . . . not only to make good steel products; but to make them still better . . . not only to fulfill today's requirements; but to anticipate tomorrow's—these are the principles that constantly guide CF&I."

G. F. Franz
President

Grinding Mill Bulletin #1

CF&I is firmly convinced of the value—to us as well as to our customers—of our president's words, quoted above. In line with this principle, we are now commencing a series of ads designed to add important technical facts to the mining industry's store of knowledge on more efficient grinding mill operations. We have purposely restricted the subject matter of these ads so that they will be of interest to only one segment of the readers of this magazine—grinding mill operators. And this is right in line with CF&I's policy—"*not merely to sell; but to serve.*"

Grinding Ball Rationing of the Makeup Charge

A practical means of improving many grinding mill operations is to determine the optimum size assortment of grinding balls that should be added as a makeup charge.

Increased Milling Throughput Reduces Milling Cost

Total milling cost will *decrease* as number of tons processed (throughput) *increases*. This advantage can be realized in any plant, provided that the rate of mining and all steps in ore processing can be increased to match the greater throughput of the ball mill, which makeup charge rationing makes possible.

Indications That Rationed Charge Is Needed

When a one-size ball addition to a mill is being established, the following conditions may indicate that ball rationing is warranted:

1) There may be a certain amount of tramp oversize that can be reduced by replacing a portion of the balls by a larger size.
2) There may be a crowding of particles of reduced size but not of finished size, showing a deficiency of small-size balls.

When a new mill is started up, time is needed to get the process running smoothly. To add a further problem of determining a makeup ration during this start-up period would be ill-timed.

Ball Wear Pattern

When balls of one size are used for addition, the seasoned charge in the mill ranges from balls of the original diameter to those small enough to purge from the mill. If a screen analysis of the ball charge is made using screens with openings of equal increments, such as ½ inch, the weight of the balls on each screen will show a certain pattern or distribution of the charge by weight. Apparently the rate of wear of different ball sizes in the charge is affected by the size structure of the mill feed and by the crystal size of the minerals. Also, the physical and metallurgical characteristics of the ball may vary with the distance from its center.

There is evidence that there is a difference in the wear pattern of grinding balls of different manufacture. Some appear to be *self-rationing* while others do not. This self-rationing quality, the built-in ability to wear evenly and give longest possible service life, is an important feature of CF&I Forged Steel Grinding Balls. Proper analysis steel, and continuous control and inspection at every stage of production help CF&I to impart this self-rationing quality to its grinding balls.

Impact, Nipping and Attrition Grinding

Large balls in the charge drop with greater impact and also have a more effective nipping action on the larger particles. Small balls make a greater number of contacts (since there are more of them), so that attrition grinding is more effective. Fine crushing is increased due to greater nipping action incidence on the small particles. Ball rationing is employed to change the size distribution of the ball charge to one that has a better ratio of impact, nipping and attrition.

Not All Mills Should Attempt Ball Rationing

Most small mills, and some of the larger ones, do not have the facilities to blend mine run ore for mill feed. With a great variation in character of feed, establishing even an optimum single-size ball charge may be a difficult problem. Facilities, personnel, and sufficient time may not be available for careful testing. The cost of such an effort may not be justified. Total savings in dollars and cents, through a relatively minor improvement in grinding practice, will not be as great in a small operation as in a larger one. Nevertheless, it appears obvious that, for many operations, the attaining of an ideal ball ration can increase the efficiency of milling sufficiently to warrant the effort involved in its development. Other pertinent factors will be discussed later in this series.

For a reprint of the article on which this ad is based, please write on your company letterhead to: Mining Supply Department, The Colorado Fuel and Iron Corporation, P. O. Box 1920, Denver, Colorado.

OTHER CF&I STEEL PRODUCTS FOR THE MINING INDUSTRY

CF&I Grinding Rods · CF&I Grader Blades · CF&I Industrial Screens
CF&I Mine Rail and Accessories · Wickwire Rope · CF&I Rock Bolts

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THE COLORADO FUEL AND IRON CORPORATION

5822

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RIPPING DIGGING

This CAT No. 977 Traxcavator increased production 30%

Dixie Mines, Inc., one of the nation's largest producers of hi-grade mica, was using a competitive machine until last year in one of their mines. This mine, near Heflin, Alabama, is under the direct operating supervision of the firm of Bailey and Van Horn. Then a Caterpillar No. 977 Traxcavator with No. 6 Ripper was put to work. Result:

"By using the No. 977 we have upped our production 30%," says Joe W. Bailey, resident manager. "It loads faster, lifts faster and is mobile. Besides working the overburden and the mica bearing pegmatite, we use it to feed the mill at night, build dams, maintain roads and many other jobs. It's a good all-around machine—and we're happy with the freedom from down time and low cost of operation."

The No. 977 rips the tough pegmatite then loads it into trucks. During a ten-hour day, production averages 500 cu. yd. The material is put through a jaw crusher, washed and screened, leaving a concentrate of mica as residue.

The No. 977 works at such a money-making pace because it's built from the ground up as a loader-excavator, not just a tractor with bucket attached. About 26 sq. ft. of ground-gripping track provide tremendous traction to crowd the 2¼-yd. bucket full every time, get away fast.

In addition, operator comfort means that this fast pace is maintained. He has a high seat with excellent visibility of all bucket positions. Controls are easily reached—bucket control a one-hand operation.

Your Caterpillar Dealer has three sizes of Traxcavators in his line. Get the full story on them today. Have him demonstrate on your job. Count on him for fast service and quality Caterpillar parts.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

CATERPILLAR

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THE NO. 977—POWER AND VERSATILITY FOR MINING

PIMA: a three-part story

Geology
Open Pit
Milling

The first AIME report on activities at the Pima mine was presented in the February 1954 issue of Mining Engineering. That article covered events leading to discovery of the orebody by geophysical methods and described the exploration and development program that eventually led to the present producing stage of the mine. At the time that account was presented, plans were under way to examine the advisability of mining the deposit by open pit methods. The following article carries the story of the Pima mine from 1954 to January 1958.

Contributed by
R. E. Thurmond
J. F. Olk
G. A. Komadina
J. A. Journeay
E. D. Spaulding
R. W. Hernlund

Pima orebody was discovered in 1950; extensive underground development was begun in 1952. In August 1954 the parent company, Union Oil Co. of California, granted Cyprus Mines Corp. an option to examine the property, and Utah Construction Co. was engaged to study the economic possibilities of mining by open pit.

After sampling and drilling to check the work completed under the original Pima management, Cyprus purchased a three-quarter interest in the property, Union Oil retaining one quarter. Cyprus later sold a one-quarter interest to Utah Construction Co., retaining half interest and management responsibility.

First ore was reached by stripping October 1956, and first concentrate was produced in December.

The mine lies about 20 miles southwest of Tucson, Ariz. Some 250 people are now employed, all of whom live in Tucson and commute daily.

The Pit

THE Pima pit is a 1700x1400-ft oval, the long axis parallel to the strike of the orebody. The north side of the pit is carried as a final pit slope that coincides with the footwall of the orebody. The south side and east and west ends of the pit are working slopes continually being stripped back toward the final slopes.

An inclined roadway extending from the natural ground surface on a 5 pct grade down to the northeast corner provides access to the pit. This road enters the pit 130 ft below the natural surface and continues as a pit ramp on a 5 pct grade to the 3150-ft bench (roughly the base of the alluvial cover). At this point the ramp system is steepened to a 12 pct grade and continues to the pit bottom. The 5 pct grade is maintained in the alluvial section to facilitate scraper haulage out of the pit. In addition to this main access ramp, temporary working ramps in the alluvial areas allow shorter routes to dumps. These are left on top of working benches and do not change the overall working slopes. Below the base of the alluvium, haulage is by truck down to the skip loading point or up from the bottom to the skip loading point.

The final pit slopes in the alluvium are laid out at 1.2:1 overall with 50-ft bank heights, 0.6:1 bank

R. E. THURMOND is Mine Superintendent, Pima Mining Co., Tucson, Ariz.; J. F. OLK is Chief Mining Engineer; G. A. KOMADINA, Mill Superintendent; J. A. JOURNEAY, Geologist; E. D. SPAULDING, Resident Manager; and R. W. Hernlund, Metallurgist.

Rockover Skip System

Two views of drive over loading structure for 38° inclined skip system.

slopes, and 30-ft benches except at the base of the alluvium, where a 50-ft bench was left as protection against excessive sloughing. Final slopes in the rock are laid out at 1:1 with 40-ft bank heights, 0.375:1 bank slopes, and alternate 10 and 40-ft bench widths.

Working slopes in the alluvium are maintained at 1.35:1 with alternate 25 and 50-ft benches. Bank slopes and heights are the same as final slopes. Working slopes in the rock are usually held at 2:1 overall with 50 to 60-ft benches, 40-ft bank heights, and approximately 0.5:1 bank slopes.

An incline for the skip hoist trackage was left on the center of the north (final) slope. Slotting into the pit slope on the upper benches and allowing a slight protrusion on the lower benches permitted a 38° skipway incline—somewhat flatter than the overall final slope.

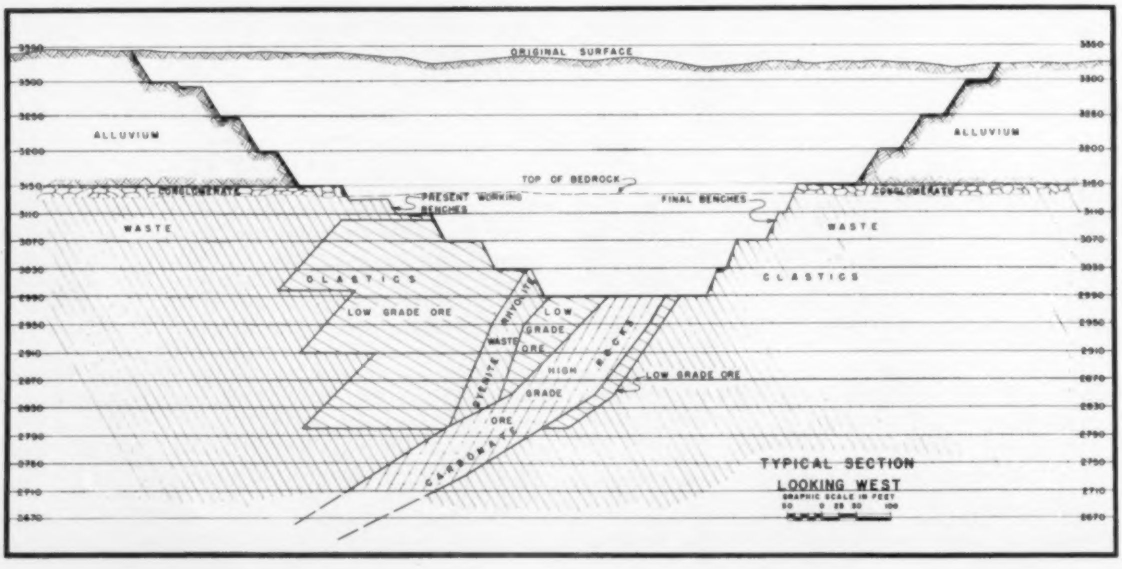
Pre-Mine Stripping: Utah Construction Co., which had been awarded the pre-stripping contract on the basis of low bid, started actual stripping in November 1955. MRS tractor units and Wooldridge 34-cu yd scrapers were used in conjunction with one Marion 151-M shovel and four LLD Euclid trucks. Utah stripped approximately 6 million cu yd during its contract (Nov. 1 to Oct. 1, 1956). Pima commenced stripping operations alongside the contractor in April 1956 and by the end of that year had moved about 3 million cu yd. During stripping, Pima trained a competent group of employes to operate the pit after Utah completed its contract.

Total pre-mine stripping amounted to a little more than 9 million cu yd. About 1 million cu yd of this was rock and the remainder alluvium.

At the present time stripping rate is about 3.0 cu yd of waste per ton of ore. Rate for the remaining life of the mine will be about 2.6 cu yd of waste per ton of ore.

Production

Daily mine production is set at approximately 4000 tons of ore on the basis of a 6-day week. Ore is mined on one shift and rock stripped on the other two shifts. Alluvium is stripped on all three. Normal payroll for the pit is 114, including supervision. Daily



production averages 12,000 cu yd—1700 cu yd of ore (4000 tons), 4500 cu yd of waste rock, and 6000 cu yd of alluvium. Output is therefore approximately 107 cu yd per manshift.

Between Nov. 1, 1955 and Jan. 1, 1958, 12.8 million cu yd of alluvium, rock, and ore were moved from the pit. Of this total excavation, 5.9 million cu yd were moved by a contractor as pre-mine stripping. A total of 1.1 million tons of sulfide ore averaging 1.74 pct Cu has been handled by the concentrator. In addition, 659,668 tons of mixed sulfide and oxide ore have been stockpiled.

Alluvial Stripping: Nearly all the alluvial stripping is being done with scrapers. Six Caterpillar DW-21's, two D-8 Pushcats, and one D-8 dozer comprise the dirt-moving equipment. Haul roads are maintained by a Caterpillar No. 12 motor grader and a 3000-gal GMC water truck. These units also maintain truck haul roads in the pit and on surface.

Single-unit push loading is used, with downhill loading on 10 to 20 pct inclines. An Ateco ripper is mounted on one of the Pushcats and an effort is made to keep the scraper well ripped. Water is flooded onto the cut wherever possible to lay dust and improve loading conditions.

Since May 1957 four scrapers have been scheduled for each of three shifts, and two remain in the shop for servicing and maintenance. About 6000 cu yd of alluvium are being moved per day.

Drilling and Blasting: Primary drilling is done with one 40-R Bucyrus-Erie rotary drill working two 8-hr shifts per day. Drilling averages 350 ft of 9-ft hole per shift. Hughes W-7-R Aerotype tricone bits are used almost exclusively. Average bit life is about 2500 ft and individual bit life varies from 1500 to 4000 ft, depending on the rock type being drilled.

From a drilling and blasting standpoint, there are three types of rocks in the pit:

1) A very dense, nonfractured conglomerate that is easy to drill but very difficult to fragment.

2) A highly fractured brittle hanging wall rock (quartzose-clastic series), which is medium hard drilling and very easy to fragment.

3) An altered carbonate rock (host rock for high grade mineralization), which is very hard drilling relative to the degree of sulfide mineralization and somewhat difficult to fragment. Drillhole spacing and loading varies for each type and different blasting agents are used in some cases.

The *conglomerate* is generally worked in two 20-ft lifts in order to reduce toe burden and facilitate working the banks with shovels. Holes are drilled on 10 to 12-ft centers with 3 ft of subgrade drilling. Toeholing is a normal practice in all conglomerate shots. Holes are loaded with high-strength gelatin base dynamite. Deck charging is usually employed, the charge being placed in the more dense conglomerate layer as indicated in the bank face. Calculated powder factors will range from 1.2 to 1.6 lb per cu yd, depending on conditions. Fragmentation is reasonably good, but considerable secondary blasting is required to size material for handling through the skip hoist system.

The *hanging wall rock* is drilled with much wider spacing—from 20 to 24 ft is normal. Three feet of subgrade drilling is usual for 40-ft banks. Normal crest burden is 10 ft; normal toe burden 24 ft. This rock is shot with ammonium nitrate blasting agent, using about 20 pct high strength gelatin base dynamite as a priming agent. Powder factors range from

0.5 to 0.7 lb per cu yd, based on calculated burden. Fragmentation is invariably good—little secondary work is necessary.

In the *altered carbonate rock*, blastholes are generally spaced from 15 to 18 ft. Subgrade drilling in this rock is increased to 5 to 6 ft for a 40-ft bank. Normal crest burden is 10 ft, normal toe burden 25 ft. Longer toes must be toe-holed; however, this is necessary only occasionally. Blastholes are loaded with ammonium nitrate blasting agent primed with 20 pct high-strength gelatin base explosive. The calculated powder factor varies from 1.0 to 1.2 lb per cu yd. Fragmentation is not always good, and some secondary blasting is required. All primary shots are connected with Primacord and detonated with black fuse and a blasting cap. Millisecond Primacord connectors are often used to improve fragmentation and control backbreak. Both mudcapping and blockholing are used for secondary blasting. Blockholing is favored whenever time is available for the drilling.

Skip Hoist System

Three Bucyrus-Erie 54-B (2½-cu yd capacity) diesel-powered shovels load waste rock and ore. These small shovels were selected for several reasons:

1) Mobility is essential because shovels must be moved often from high grade to low grade ore, or to waste.

2) Two shovels are usually needed simultaneously in ore for blending, and small shovels were required to fit production schedules.

3) Small buckets were needed to accomplish sizing of material for the skip hoist system. Two shovels are scheduled on each of the three shifts, and one shovel is idle for maintenance on each shift. Average production is 1000 cu yd per shovel shift.

Eleven Kenworth trucks and a National Iron Rockover skip hoist system transport the ore and waste rock. Seven of the trucks are 16-cu yd end dumps powered by 335-hp (NRT) turbocharged Cummins engines with Allison Torquematic transmissions. These units are used in the pit for hauling from the shovels to the skip loading pocket. The remaining four trucks are 34-cu yd end dump tractor-trailers powered by the same engine but with a 10-speed Fuller transmission. These units are used on the surface to haul from skip headframe to crusher and to the waste dump. Normally five of the 16-cu yd end dumps are scheduled in the pit, servicing the bottom of the skip hoist system. Three of the 34-cu yd tractor-trailers handle surface hauling away from the receiving hoppers.

The skip headframe and hoist are on the north slope of the pit at the 3316 elevation and the skip loading structure is just below the 3070 bench. Additional stations will be installed as the pit becomes deeper.

At this time, ore and waste rock are being mined on four benches—the 3110, 3070, 3030, and 2990. Material is hauled down two benches or up two benches to the 3070 loading pocket by the smaller pit trucks. Then it is hoisted—a vertical distance of 291 ft—over the skip railway and dumped into the receiving hoppers. From this point it is loaded into the tractor-trailer units and hauled to the crusher or waste dumps.

Skip vs Truck: The skip hoist system was chosen over truck haulage primarily to reduce stripping requirements and operational costs. Trucking ore and waste out of the pit would require establishing permanent haul roads at least 50 ft wide with a maximum 8 pct grade. To reach the lower limits of the pit, these roads would require a minimum of 100 ft of additional stripping at the surface beyond the stripping limits established to recover the ore.

With a skip hoist system, temporary roads could be cut where necessary to haul from one bench to another for access to the skips. Service into the pit would require only a 12 pct road 30 ft wide.

At the time pit studies were being made, four skip hoist systems were in operation in the Mesabi Iron Range. Investigation of these operations and the cost information obtained pointed up the following advantages of using a skip hoist system:

1) Low maintenance cost.
2) Low operational cost.
3) Reduction in haul road maintenance.
4) Low investment in spare parts and stand-by equipment.
5) Minimum stripping required for haul system.
6) Decreased maintenance for trucks.

Operating cost estimates were made for an all-truck operation and for a skip hoist system in conjunction with trucks. For the truck operation, it was estimated that 22 16-cu yd trucks would be required. A saving of 7¢ per ton was indicated with the skip hoist system, but initial capital outlay would be greater. It was calculated that this additional capital would be amortized in a reasonable time by the savings in operational costs. For the Pima pit, from nearly every standpoint, the skip hoist system seemed preferable to all-truck haulage.

Hoist and Headframe: The skip hoist railway consists of a double set of 10-ft 2-in. gage tracks separated by 3 ft 6 in. The tracks converge slightly at the base of the headframe and continue up the headframe to permit dumping into two 200-ton receiving hoppers that are an integral part of the structure. Two hoppers are required so that both ore and waste may be handled simultaneously. Through use of gates actuated by hydraulic cylinders and operated from the hoist control console, skip loads can be sent into either hopper by the hoist operator.

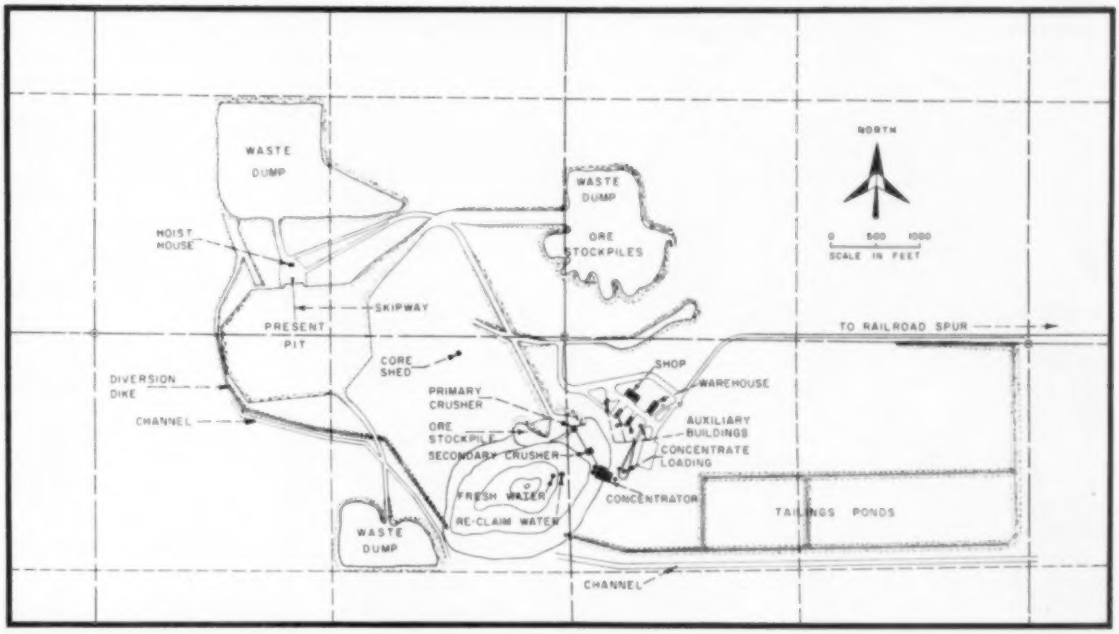
The receiving hoppers are discharged by reciprocating plate feeders, actuated by hydraulic cylinders. These feeders will load a 34-cu yd truck in 90 sec.

A steel skip loading structure bridges the slot for the skip trackage where it cuts through the 3070 bench. The skips are spotted below the bridge under loading hoppers, which are an integral part of the bridge structure. Decking consists of two steel panels, supported on wheels that travel on tracks.

Deck panels are opened and closed by compressed air cylinders. After a truck has driven over the decking and is spotted in dump position, the decks are opened and the load is dumped into the loading hopper and thence into the skip.

Installed only recently, this system replaces a temporary loading structure—an excavation about 18 ft below the bench with the track continued to the bottom. Trucks on the bench could dump directly over the sides of the skips. A simple muck-deflecting structure was installed at the loading station to position the load properly in the skips.

The 22-ton skip chassis is essentially a rectangular open frame with a spring-suspended wheel at



Geology

Bedrock is overlain by 200 ft of alluvial wash, which has a regular eastward slope, and there are no outcroppings within a 1500-ft radius of the mine. Lying immediately above bedrock is an irregular conglomerate zone (0 to 25 ft thick) of medium to coarse texture, composed of igneous and sedimentary rocks cemented by siliceous and calcareous material.

The Pima orebody is a pyrometasomatic (contact metamorphic) deposit. The main ore zone has an average dip of 45° to the south and trends east-west, curving from a northwesterly bearing on the west to a slightly northeasterly direction on the east. The ore zone is variable in thickness, probably averaging 200 ft, and this zone has been developed over a lateral span of 1600 ft in the main part of the mine. It extends into a neighboring property on the west, but to the east geology and mineralization are obscure. The lower limits of the zone have not been determined, but it has been intersected by drillholes at about 800 ft vertical depth. Paralleling the main ore zone on its footwall side is a persistent breccia zone, which extends to the northeast beyond the present known main orebody.

Three Rock Classes

Determination of age and identification of rock types has been complicated, since most of the rocks are moderately to highly altered or metamorphosed. Many of the less altered rocks are fine-grained; consequently field determinations have often been uncertain, and even petrographic studies have not been definite. The rocks fall into three broad classes: 1) carbonate, 2) clastic, and 3) igneous.

The carbonate rocks, grouped under the term *hornfels*, constitute the main high grade ore zone. These rocks are garnet (Grossularite) hornfels, diopside hornfels, and tremolite hornfels. Dolomite and limestone are present in varying amounts. The main ore sulfide is chalcopyrite, with minor amounts of chalcocite, native copper, chrysocolla, tenorite, bornite, and cuprite.

The zone of oxidation extends erratically 40 to 50 ft below top of bedrock. The chalcopyrite forms local and highly irregular concentrations, and shows a tendency to favor one type of hornfels more than another.

Basis of Open Pit Operation

The clastic rocks, occurring in both hanging and footwalls of the carbonate formation, are extremely fine-grained with a quartzitic appearance. They are composed of quartz, feldspar, and sericite, and texture is definitely clastic. In some places it is almost sedimentary; in other it has an igneous appearance. On the basis of petrographic work and visual examination of drill core, those at Pima believe that some of the clastic rocks formerly called *arkosite* may be better classified as pyroclastics. Even though they cannot be decisively proven petrographically, these pyroclastics may contain local accumulations of sediments, clastic and otherwise. In the hanging wall clastic rocks, pyrite is widely disseminated and there are zones of low grade disseminated chalcopyrite mineralizations, which makes an open pit operation feasible.

The igneous rocks found at the mine are of intrusive nature and consist of rhyolite, syenite, and quartz monzonite porphyry. The rhyolites and syenites are unmineralized and occur in and above the hanging wall of the carbonate series. The bulk of the quartz monzonite porphyry is found in the footwall clastics and is slightly mineralized by pyrite and chalcopyrite.

each corner. The rear axle is mounted to pivot centrally between the rear wheels, providing three-point suspension for the chassis and permitting the skips to traverse relatively uneven track at high speeds without derailment.

The skip body is carried within the skip chassis by a trunnion on each side at about the center of gravity of the load, so that dumping the skip is merely a rocking action of the body within the chassis. The skips are dumped by means of scrolls on dump plates carried adjacent to the skip track on the headframe and above the receiving hoppers. A dump wheel is mounted at the lower forward end of each side of the skip body. As the skip travels into the dumping plates, the body latches are first automatically released by a cam on the dump plates; then the dump wheels on the skip body are carried through the scroll of the dump plates to rotate the body at the proper rate and discharge the load.

The skip chassis and body are welded double-wall construction employing a network of diaphrams separating the two walls. This provides shock-absorbing qualities for loading directly from trucks.

Electrical Features: The skip hoist consists of two cylindrical drums coupled by an expanding toggle-type clutch engaging an internal geared ring. The 7.8 x 8.2 ft diam drums are grooved for 1⅞-in. hoist rope. The hoist is powered by four 500-hp, 1750-rpm, 4160-v, three-phase, 60-cycle, wound rotor motors. Power is transmitted to the drums by two twin pinion gear reducers. Each has two input shafts from two motors and one output shaft to the drum. The gear ratio is 27.8 to 1.

In addition to reverse current braking, there is positive braking by four spring-set shoe brakes, one at each motor coupling. Each brake affords 4000 ft-lb torque. Brakes are released by d-c solenoids, supplied by Rectox rectifiers.

Primary power control is full magnetic quadruplex reversing-plugging with a five-position reversing master switch. Thus the operator has hoist acceleration control (eight steps with five under control) and retarding control (two steps only) by plugging or reverse current braking. Duplex magnetic control is available for single-skip operation when skips are being adjusted with clutch disengaged. In normal operation, metered demand is 700 kw, integrated over a 15-min period. Highest peak demand is about 3200 kw. Normal power consumption averages 250 kw-hr.

Automatic operational protection is provided for overspeed, overtravel, reverse travel, and brake malfunction. Each protective device is backed by a second device in case the first functions imperfectly.

The hoist control console is located near the top of the headframe where the operator has a full view and complete control of the hoisting operation, declutching and adjustment of skips, and operation of flop gates for the receiving hoppers. Through a P. A. system he can communicate with the skip tender in the pit, the receiving hopper feeder operator, and the hoist house.

Total cycle time from the 3070 bench is 67.0 sec —acceleration, 18.0 sec; full speed (1650 fpm), 5.0 sec; deceleration, 14.0 sec; loading and unloading, 30.0 sec.

In a month of normal operation, the hoist handles 6000 cu yd per day of waste rock or ore. This is equivalent to 14,000 tons, or about 700 skips on a three-shift per day operation.

Open Pit Equipment Maintenance

Pit equipment is serviced, maintained, and repaired by the mechanical and electrical departments. Shop personnel consists of 35 men—mechanics, welders, servicemen, machinists, electricians, and supervision. Maintenance is scheduled on three shifts.

Each unit of pit equipment is scheduled for service and maintenance one shift per day. Since the hoist is operated three shifts every working day, maintenance and repair is done on Sundays. Under these conditions, equipment availability is very good. A tabulation of availability for a typical month is as follows:

Total Number of Units	Type of Unit		Units Scheduled Per Shift	Availability, Pct
6	DW-21	Scrapers	4	93.02
7	802	Dump trucks	5	95.61
4	802-B	Dump trucks	3	94.49
3	54-B	Shovels	2	97.86
1	40-R	Rotary drill	1	86.42
1		Skip hoist	1	92.72*

* Average availability for skip hoist, May through December 1957, 83.6 pct.

Availability is calculated as a ratio of actual hours worked to scheduled hours. Only lost time for mechanical or electrical reasons is deleted. If a unit breaks down and a replacement is ready immediately, no loss of availability is incurred.

Mechanical maintenance on the hoist machinery has been a minor item, but electrical maintenance has been a problem. Four motor failures have occurred. Three of these have been attributed to manufacturing problems, but the fourth has not been defined. Low voltage on the primary line during acceleration has resulted in overheating of resistance grids. The low voltage problem has been partially overcome by altering the settings of the various timing relays. This has reduced excessive peak currents but requires a longer than anticipated acceleration period.

Relocating the master control switch in the control console eliminated troublesome linkage.

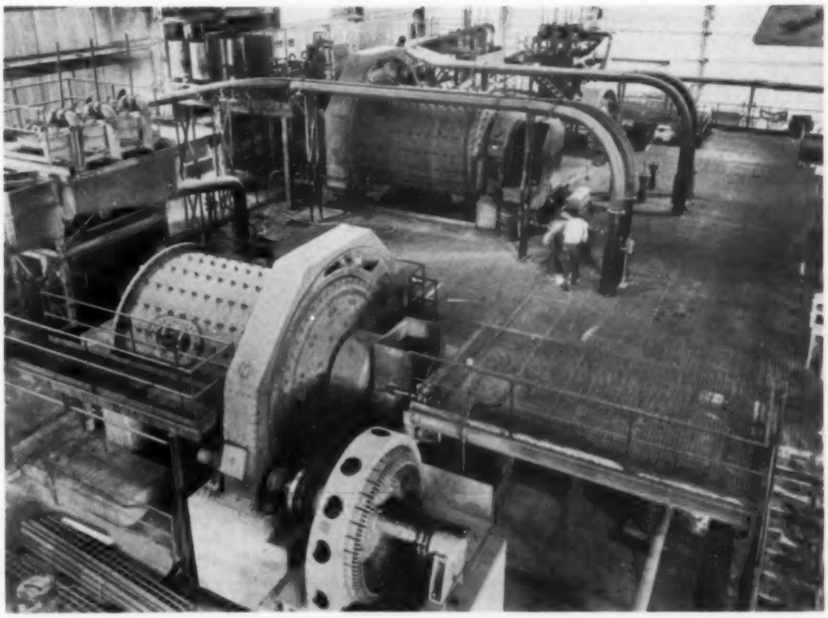
Several structural changes were made to the headframe. Where possible, rock boxes were installed at impact points to permit fine material to build up and act as a cushion. Impact rails were relocated at the top of the headframe. The main structural members of the receiving hoppers have been strengthened. The flop gates in the ore bins have had to be rebuilt to withstand the impact of falling rock but still are not satisfactory.

The skips have required reinforcement of the structural members of the body, installation of a lattice of spring steel under the liner plate, and a ¾-in. layer of medium-hard rubber behind the side plates. These changes have reduced maintenance, but more changes are contemplated to control the problem. It is believed that installation of control gates in the drive-over loading structure will materially reduce loading shock on the skip body.

At this time it is apparent that the skip system at Pima offers a definite advantage as compared to all-truck haulage, but problems resulting from direct handling of 22-ton truckloads of pit-run rock into the skips and through the headframe will require heavier equipment than anticipated.

All of the plant buildings were started in December 1955 and completed by the end of 1956. Mill production started Jan. 1, 1957, after a short run-in.

The Mill



Grinding bay of Pima concentrator shows rod mill, primary ball mills and cyclones and regrind ball mill and cyclones.

Crushing: Run-of-mine ore is fed by truck or dozer from a stockpile to the primary crusher chute. A Ross chain feeder draws ore over a 7-in. grizzly to feed a 66x84-in. Birdsboro jaw crusher set to produce −7-in. material. The crushed ore is transported to the secondary crushing and screening building by a 54-in. conveyor. The ore is screened over two double-deck Tyler F-900 Tyrock screens dressed with 2x18-in. rod grizzlies on the top deck and ¾-in. square woven wire on the lower deck. Oversize from the top decks passes to a 7-ft standard Symons crusher and oversize from lower decks to a 7-ft shorthead Symons crusher. Feed to the screens is split by an adjustable plate with a cast manganese cap. The screens and crushers are so located that cross conveyors are not necessary—the screens feed their respective products by gravity into the crushers or onto the fine ore conveyor. The crushers feed directly onto the return conveyor. At the junction of the return conveyor and the 54-in. conveyor from the jaw crusher, space has been left to permit installation of a surge bin if necessary.

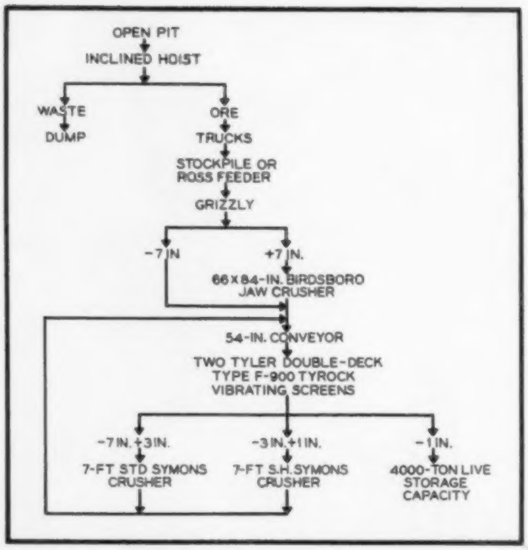
A Link-Belt automatic tripper distributes the −¾-in. ore to two fine ore bins. These bins were designed to hold 6000 live tons, but experience has proved that the unexpectedly high angle of repose (up to 70°) reduces capacity to 4000 live tons. This relatively small bin capacity for the plant, which grinds up to 3600 tons per 24 hr, has required operation of the crushing and screening plant seven days a week rather than the expected six days. It is planned to increase bin capacity enough to permit operating the mill seven days a week with the crushing and screening plant running six days.

At the bottom of each 50-ft diam steel ore bin there are parallel slots. Curved steel plates under these slots form a tapered ore feed 4 in. wide at the far end and 10 in. wide at the near or discharge end. Belt conveyors running under the slots discharge onto a collector conveyor. Space between the steel slot and the conveyor can be adjusted to insure uniform feed from the entire length of the slot.

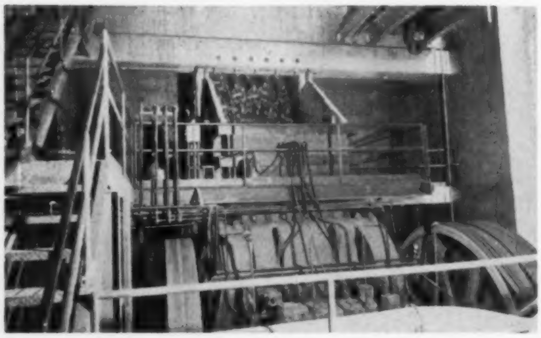
Grinding: Mill feed at rates up to 3600 tpd is drawn from any combination of two of the four belt feeders. The feeders, integrated with a weightometer to provide uniform flow rates, discharge to the spout of a 10x13-ft Allis-Chalmers rod mill operating in open circuit. The rod mill discharge is split between two primary cyclones operating in open circuit. The rod mill discharge is split between two primary cyclone feed sumps. One side utilizes a 10-in. type T Amsco dredge pump, while the parallel unit is a type CT Barrett-Haentjens solids-handling pump. Each unit delivers rod mill discharge at 50 to 57 pct solids to two 20-in. model D-20-B Krebs cyclones operated with a feed inlet pressure of 8 to 10 lb. At the present time 6¾-in. vortex finders and 3½-in. apex pieces are used.

Cyclone Separation: The cyclones are able to take a fairly wide range of feed rates and still maintain overflows of 27 to 31 pct solids. The underflow, at 80 to 85 pct solids, is delivered by a spout to a 10½x13-ft Allis-Chalmers overflow ball mill. The ball mill discharge, at 76 to 82 pct solids, returns to the cyclone feed sumps. A portion of the cyclone overflow is bled off to maintain constant levels in the sump boxes. The remainder is reduced to 27 to 28 pct solids prior to flotation with fresh water.

The regrind circuit consists of a 7x12-ft Allis-Chalmers overflow ball mill operating in closed cir-



CRUSHING FLOWSHEET



Primary Birdsboro jaw crusher below Ross chain feeder.

Secondary crushing plant with, left, 7-ft Symons shorthead, right, 7-ft Symons standard. Vibrating screens above.

cuit with six 10-in. model D-10-B Krebs cyclones. The cyclone overflow is used as dilution water in the primary grinding circuit. The use of the regrind circuit has been deferred until a change in character of ore requires finer grinding of either the middling fraction or the rougher concentrates. During 1957, 88 pct by weight of the middling product was −200 mesh and contained 85 pct of the total copper in this product.

The rod mill, operating at 15.55 rpm, is charged with 3½-in. rods at a rate of 0.444 lb per ton for 1957. Manganese steel liners are single wave type.

The primary ball mills, operating at 15.1 rpm, are currently being charged a rationed make-up of 50 pct each of 1½ and 2-in. balls at a total rate of 0.816 lb per ton for 1957. Double wave manganese steel liners are used in the ball mills. The 1,094,559 tons of ore treated in the grinding mills during 1957 caused only slightly noticeable liner wear.

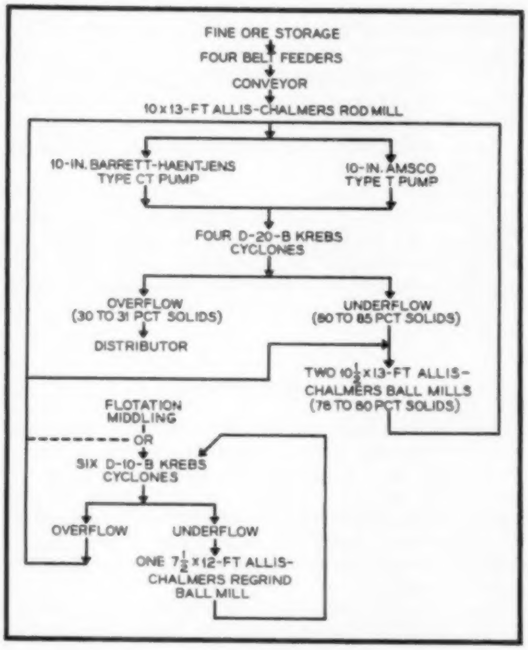
As yet there is no information to show the effect of feed rates, pressures, and other variables on classification performance of the cyclones.

The Flotation Section: The flotation feed passes through a distributor that divides the pulp to three parallel banks of 66-in. single overflow Fagergren flotation machines. The rougher concentrate from the first four to seven cells goes to the cleaner machine; the cleaner concentrate goes to the recleaner machine. The cleaner and recleaner tailings are combined with the scavenger concentrate produced on the last three to six cells of the rougher bank. At present this middling pulp is being pumped to the head of the primary grinding circuit for use as dilution water. The alternate flow is to send the middlings to the regrind circuit.

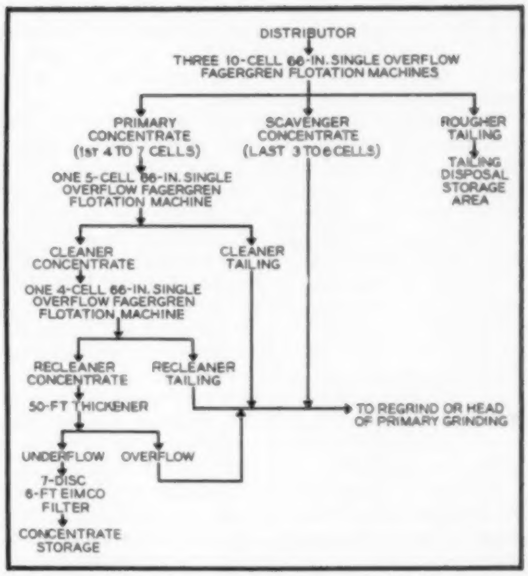
The mill layout provided for doubling rougher flotation capacity. Pilot plant tests prior to mill construction indicated that the extra capacity might not be necessary, but plant-size tests have demonstrated that the additional capital cost can be recovered in relatively short time by increased copper recovery. It is expected that total copper recovery will increase by as much as 2 pct with the increased capacity.

Recleaner concentrate flows to a 50-ft Dorr thickener, the underflow is moved by centrifugal pump to a 6-ft diam 7-disk Eimco filter, and overflow is returned to the middling sump for use as dilution water in the grinding circuit. Experience has shown that the pump can be directly connected to the underflow discharge spigot. Currently the thickener is also used for storage to permit filtering for 4 hr and then an idle storage period for 8 hr. This cycle is repeated twice every 24 hr. The filtered concentrate (containing 10 to 11 pct water) is conveyed to storage and then scraped by a Joy slusher into 25-ton Kenworth dump trucks. The concentrate is weighed and sampled for moisture at the mill and then hauled seven miles to the railroad. After the car is loaded and leveled, it is pipe-sampled in a grid pattern for copper analysis. Concentrate is shipped to the copper smelter at El Paso.

Tailing from the concentrator is pumped unthickened to slime disposal ponds 3000 ft from the mill. No tailing pond has been built from the contained sands, but this is now being considered. The reclaimed water passes through decant towers to a gathering dump, from which it is pumped to the 250,000-gal mill water storage tank above the mill. The mill water is treated with phosphate to minimize lime scaling at the discharge of this storage.



GRINDING FLOWSHEET



CONCENTRATION FLOWSHEET

Flotation Concentrate Assayed Screen Analysis, May Through December 1957

Size	Percent Retained	Cumulative Percent Retained	Assay Total Copper, Pct	Distribution: Percent	Distribution: Cumulative Percent
+100	1.71		13.36	0.93	
+150	4.60	6.31	17.30	3.24	4.17
+200	12.67	18.98	21.87	11.27	15.44
+270	12.91	31.89	23.85	12.52	27.96
+325	8.50	40.39	23.88	8.25	36.21
−325	59.61	100.00	26.31	63.79	100.00
Total	100.00		24.59	100.00	

Flotation Tailing Assayed Screen Analysis, May Through December 1957

Size	Percent Retained	Cumulative Percent Retained	Total Copper, Pct	Distribution: Percent	Distribution: Cumulative Percent
+65	1.94		0.40	3.18	
+100	6.50	8.44	0.33	8.71	11.89
+150	8.42	16.88	0.32	11.16	23.05
+200	12.81	29.67	0.28	14.58	37.63
+270	9.66	39.33	0.23	9.29	46.92
+325	6.21	45.54	0.22	5.67	52.59
−325	54.46	100.00	0.21	47.41	100.00
Total	100.00		0.24*	100.00	

* Of this figure 0.18 pct is nonoxide copper and 0.06 pct oxide copper.

Rod Mill Screen Analysis, 1957

Feed: Size	Feed: Percent Retained	Feed: Cum Pct Retained	Discharge: Size	Discharge: Percent Retained	Discharge: Cum Pct Retained
+1 in.	8.65	—	+6	0.82	—
−1 +½ in.	25.17	33.82	+8	1.81	2.63
−½ in. +4 mesh	30.00	63.82	+10	3.92	6.55
−4 mesh	36.18	100.00	+28	26.66	33.21
Total	100.00		+48	15.80	49.01
			+65	5.42	54.43
			+100	6.12	60.55
			+150	4.49	65.04
			+200	5.42	70.46
			+325	6.32	76.78
			−325	23.22	100.00
			Total	100.00	

Summary of Operating Data, 1957

Cyclone overflow: 11.51 pH
Rod consumption: 0.444 lb per ton
Ball consumption: 0.816 lb per ton
Crusher operated (single-shift basis): 82.10 pct
Mill operated (three-shift, seven-day basis): 93.83 pct
Ore: 2.928 sp gr

Solids in Mill Pulps	Pct
Rod mill discharge	68.26
No. 1 cyclone feed	52.28
No. 1 cyclone underflow	82.44
No. 1 cyclone overflow	28.93
No. 1 ball mill discharge	76.78
No. 2 cyclone feed	53.84
No. 2 cyclone overflow	81.48
No. 2 cyclone underflow	30.11
No. 2 ball mill discharge	76.43
Rougher flotation feed	27.32
Cleaner flotation feed	17.84
Recleaner flotation feed	17.39
Thickener underflow	51.79

Metallurgical Summary, 1957

Product	Dry Tons	Assay, Pct: Total Cu	Assay, Pct: Oxide Cu	Assay, Pct: Sulfide Cu	Distribution, Pct: Total Cu	Distribution, Pct: Oxide Cu	Distribution, Pct: Sulfide Cu
Mill Feed	1,094,559	1.74	0.10	1.64	100.00	100.00	100.00
Copper Concentrate	67,288	24.28	0.32	23.96	85.78	19.14	90.00
Tailings	1,027,271	0.26	0.09	0.17	14.22	80.86	10.00

Fresh water is pumped 6 miles from wells along the Santa Cruz river to a 50,000-gal storage tank. This is the potable water supply as well as make-up water for the mill. The mill requires 240 gal of make-up water per ton of ore.

Reagents: Mill reagents are mixed on day-shift and pumped to day storage tanks. Pebble lime is purchased and slaked in a Dorr lime slaker. The lime slurry is pumped throughout the mill and withdrawn as needed. Nearly all the lime being added is used in the grinding circuit. A pH of 11.2 to 11.6 has been found optimum for current ores.

Choice of reagents to be used has not been fully made, and mill testing will be continued for several more months before this is determined. Lime, Z-6 and/or Z-11 xanthate and methyl isobutyl carbinol (MIBC) are added to the rod mill. Z-6 and Z-11 xanthate and methyl isobutyl carbinol frother are added to the distributor, the second, fourth, sixth and eighth cells. Sodium sulfide, when needed, is added to the fourth, sixth and eighth cells.

During early months of operation Minerec A and B, Reagent 404, Dowfroth 250, and Z-11 were also used in the flotation circuit. As more mill testing was completed, it was found that reagent modification could be made. The average reagent consumption in pounds per ton of ore milled during the last six months of 1957 was as follows:

Lime	7.128
Aerofloat 25	0.009*
Dowfroth 250	0.047*
MIBC	0.109
Z-6 xanthate	0.055
Z-11 xanthate	0.041
Sodium cyanide	0.015**
Sodium sulfide	0.311
Phosphate	0.017

* Discontinued use in early September.
** For December only.

The concentrator is laid out to utilize the natural slope for gravity flows through the plant. The entire plant is designed to use minimum labor. The crushing crew, working on single shift under a foreman, consists of one primary and one secondary operator and two laborers who clean up spills and remove wood and steel from the ore. Steel is removed by hand—a tramp steel detector will not work because of the magnetite content of the ore.

While the plant is operating on a seven-day basis, a relief operator replaces the two who now work a five-day week. The concentrator is handled by a shift foreman, a grinding operator, and a flotation operator. On day-shift a concentrate loader and reagent man are also available. Maintenance of the mill and crusher is handled by nine repairmen, one oiler, and three laborers under the direction of the repair foreman. The total mill manpower of 46 includes supervision, operation, maintenance, metallurgy, sampling (but not assaying), and clerical.

Electrical power purchased from the Tucson Gas, Electric Light & Power Co. is delivered to the Pima substation at 44,000 v. Power is stepped down to 4160 v for distribution to the mine and mill. Power consumption for the year per ton is as follows:

Crushing	0.876
Concentrator	15.453
Fresh water	2.392
Tailing return water	0.341
Pit	1.650
Shops and miscellaneous	0.463
Total	21.175 kw-hr per ton milled

During 1957 a test was conducted using two different makes of pumps to deliver rod and ball mill discharge to the primary cyclones. Results to date have indicated a cost of $0.70 per hr for maintenance supplies and labor for each pump. A rubber-lined pump will be used experimentally.

Only slight wear is revealed on the cyclones and on the rubber-lined pipe from the cyclone feed pumps to the cyclone feed manifold. Fixed rubber apex pieces lasted 18 days as compared to a minimum of 120 days for the ceramic apex pieces now being used. Some of the ceramic type that have been in service more than 150 days are still satisfactory, so the average life will be high as compared to rubber.

Engineering

The engineering department works in close cooperation with the production departments and is responsible for operating layouts, estimates, pit schedules, and uniformity of mill feed. Present personnel consists of a chief mining engineer, pit engineer, ore control engineer, and draftsman and a field survey crew of three—an instrument man and two rodmen.

Mine operating layouts are prepared on the basis of a series of pit expansions. Each expansion is scheduled for completion of alluvium and waste rock stripping before the ore is exhausted in the previous expansion. Layouts are made on current pit maps on a scale of 1 in. to 50 ft. Volume estimates of ore and stripping are made on horizontal level maps on which the ore blocks are outlined. Cross sections on a scale of 1 in. to 50 ft are used in planning the layouts, although currently these are not used for estimating. Owing to the size and shape of the present pit, a more accurate estimate can be obtained from horizontal level maps.

Volume estimates of material removed from the pit are made quarterly on specially prepared section. To keep the pit map and ore control maps up to date the pit is surveyed each month, with a more accurate survey when a volume estimate is made.

For ore control, each blasthole drilled is sampled and assayed. From the established grade cutoffs, the material in each pit blast is classified and where necessary is segregated. Where two or more types of material are encountered in one blast, the cutoffs are flagged in the bank for the operators' guidance. From the assays of the ore material, a blend is established between various working faces for a constant grade of mill feed. Close cooperation is maintained between the ore control engineer and the mine operating staff.

A certain amount of mechanical drafting, construction design, and detailing is done by this department for all other departments in connection with repair and alterations of facilities and small construction work. Cost estimates are also prepared and specifications set up.

The engineering department makes claim surveys, conducts time studies on equipment, maintains records on material stockpiled from the pit, provides line and grade control for pit operations, and does sampling. Because the orebody now being worked is split between a state mineral lease and federal patented claims, careful estimates are made of ore volumes of the other material removed from each section of the property, and records are kept up regularly on these volumes.

A New Zealander Looks at

Hydraulicking Coal in the USSR

by W. B. Watson

IN the USSR considerable technical improvements have been made in hydraulic methods of mining coal. In New Zealand coal mining by these methods is still comparatively crude. The Russian techniques are closely studied in New Zealand, and this description is based mainly on translations of technical articles.

Fundamentally the methods in both countries are the same—chief developments in the USSR are the use of high pressure water jets, capable of winning all but the hardest coal without prior blasting, and hydraulic pipeline transportation where the gradient is unsuitable for flume transport.

In New Zealand hydraulic methods have been used on a small scale to mine and transport coal in underground and open pit mines for more than 30 years. The coal face is blasted by explosives and the broken coal washed by a low pressure water jet into a wooden flume. Coal and water travel by gravity flow along the flume, which extends from the coal faces to the mine portal and out to dewatering screens on the surface.

Where this is not possible because the gradient is unsuitable, the coal and water are flumed to an underground hopper situated on the dip of the working faces, and after dewatering the coal is hoisted to surface. The coal is mined by room and pillar, and hydraulic methods are applied to development headings and pillar extraction.

Hydraulic Method in USSR

A hydraulic monitor is set up near the face and the water jet directed at the coal; water and broken coal are transported along the flumes to a sump, from which the coal is transported by pipeline to the shaft and up to the surface. On surface it is dewatered and dried, and the water is clarified and returned by high pressure pumps to the monitors. The surface pumps develop water pressures up to 600 psi—approximately the discharge pressure at the monitors underground, the friction head being compensated for by increase in static head. The water is taken down the shaft in a 10-in. diam pipeline, which is reduced to 8-in. diam underground and 4-in. diam in the working places. Hydraulic mining requires a continually lengthening pipeline when a face is advancing and a continually shortening pipeline when coal is mined in retreat—pipes in working places are in 12-ft lengths to facilitate handling and have quick coupling joints so that pipe can be added or removed in 6 min.

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The monitor is similar to the type used in alluvial mining, with 360° horizontal rotation and 140° vertical movement. Nozzle diameter for winning and breaking coal is 0.67 to 0.87 in., with water consumption of 0.7 to 1.18 cu ft per sec. For washing coal into flumes the diameter is 0.75 to 1.18 in., with water consumption of 2 cu ft per sec.

The flume is a metal channel 16 to 20 in. wide and 12 in. high, capable of transporting coal up to 12-in. size. Flumes are usually laid on a 5 pct gradient, but a 3 pct gradient is satisfactory if the quantity flowing is greater than 3 cu ft per sec, and a 1 pct gradient if the bottom of the flume is lined with glass or porcelain.

The coal to water ratio aimed for is 1:2, but in practice it is nearer 1:6. Flumes deliver the coal and water to a sump where the +2.75-in. coal is screened off and reduced by a hammer crusher. From the sump it is transported through a pipeline; hydraulic ejectors are used for short lengths of horizontal pipelines and for small vertical lifts, low-pressure solids-handling pumps for longer horizontal pipelines, and high-pressure solids-handling pumps for pumping coal up shafts. The coal pipeline is 8 to 10-in. diam and has flushing points every 45 ft for clearing any blockage in the pipeline.

Experiments with higher water pressures up to 3000 psi are being made to eliminate the need for partial blasting in hard coal. Combined mechanical and hydraulic methods offer another possibility for hard coal, and a combination of coal plow and hydraulic transportation is suggested for seams less than 20 in. thick.

Development Headings: Development headings in coal are driven, where possible, on a rising gradient of 5 pct to facilitate flume transport. A monitor is set up about 6 ft from the face and the water jet directed at the weakest point to make a cut at floor level or parallel to the stratification; the cut is usually 2 to 3 ft deep but in some cases up to 6 ft. The water jet is then moved parallel to the cut and the coal is broken down into the cut. A barricade placed at floor level across the heading directs coal and water into the flume. When the heading has advanced 7 ft, the water is turned off and the heading is timbered. One monitor operator and two timber-

ers work two or three faces—while one face is being mined, timbering is carried out and the pipeline and monitor are advanced at the others. The monitor is moved up to the face every 15 to 60 ft, depending on hardness of the coal. A three-man crew usually drives 20 ft of heading per shift and occasionally more than 40 ft. Where no roof supports are set, rate of advance is 6 to 20 ft per hr, depending on water pressure and hardness of coal.

Gently Pitching Seams

Gently pitching seams are mined in panels by room and pillar methods. From the panel entries, rooms 13 ft wide are driven up the full pitch of the seam to the panel boundary to form pillars 50 ft wide and 500 ft long. In future, room width will be reduced to 5 ft to simplify the problem of roof support.

Applications in New Zealand

Fig. 1—Underground hydraulic coal mining in New Zealand.

Fig. 2—Open pit hydraulic coal mining in New Zealand.

Pillar extraction commences at the top of the panel and retreats down dip. Pillars are extracted by successive pockets 15 ft wide turned off right and left from each room and driven halfway across pillars to form a stepped pillar line. No roof supports are set in the pockets, but a coal fender 3 to 6 ft wide is left against the previously caved pocket and, where necessary, 1 ft of roof coal. This coal is mined retreating out of the pocket, but owing to prior collapse of the roof there is a 10 to 20 pct loss.

Rooms and pockets are mined by monitor and the coal is transported along flumes laid in each room; the pockets are set off from the rooms at such an angle that coal and water flow along the floor out of the pocket into the flume. If the coal is hard, it may be weakened by pulsed water infusion with holes drilled from the rooms halfway across the pillars before it is mined by the monitor. No track is laid underground, but a mono-rail is suspended from the roof for transportation of supplies.

Operating Results: In the USSR a seam of hard coal with weak roof and floor, average thickness 6 ft, gradient 10 pct, and depth of cover 200 ft, is mined by the method described. Four months after panel working commenced and while the technique was still being developed, production per manshift was two and a half to three times that of the best mines in the district using conventional methods. Production per monitor is 65 to 75 tons (short tons) per shift with a three-man crew. Overall production per manshift is reported to be 4 tons and by the end of 1956 was expected to be 5 tons. These figures are not high in comparison with American mechanized mining but compare favorably with European production, and it is further claimed that the capital cost of constructing a hydraulic mine is only 33 to 40 pct of the cost of a mine to be worked by conventional methods.

Steeply Pitching Seams

Vertical and steeply pitching seams are mined by sublevel caving. Shafts are sunk in the seam, and from the shafts, levels are driven in the seam in both directions at vertical intervals of 150 ft. At intervals of 200 to 400 ft along the levels, raises are driven in the seam to connect adjacent levels and to block out panels of coal for mining. Each panel is split by sublevels driven from the raises to form pillars of coal 20 ft thick between sublevels. For thick seams two sets of raises and sublevels are driven (one on the hanging wall and one on the footwall) and connected at intervals by crosscuts, but it has been found that seams up to 50 ft thick can be mined by one set of entries.

The panels on each level and individual panels are mined retreating towards the shaft—the coal in each panel is mined descending from the upper level to the next lower level, and pillars between sublevels are mined ascending from each sublevel to the next. A hydraulic monitor is set up in each sublevel and the water jet directed at the lower edge of the pillar; a layer of coal up to 10 or 15 ft wide over the full thickness of the seam is broken down under the action of the water jet and the natural tendency of the coal to cave. This is continued upward until there is a breakthrough to the caved rock in the upper sublevel. No supports are set during pillar extraction, but the top 1½ to 3 ft of coal (or bridge) is maintained as long as possible by breaking down the coal in a definite sequence, both up to the bridge and across the seam. This facili-

USSR uses both hydraulic mining and transport.

Fig. 3—Underground hydraulic monitor in the USSR.

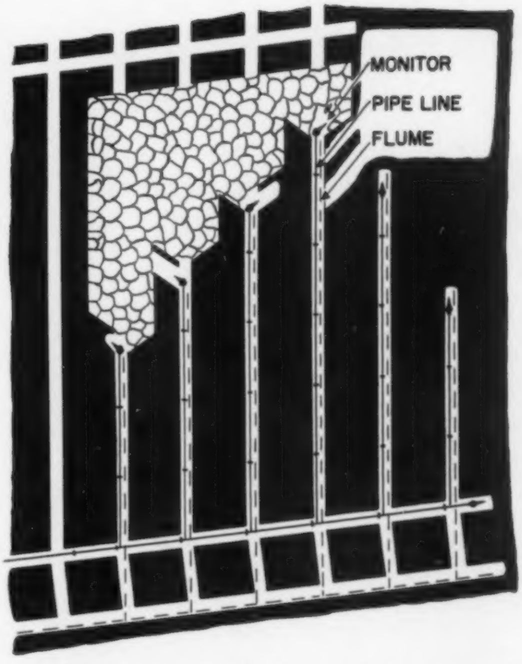

Fig. 4—Underground hydraulic coal mining in the USSR. Gently pitching seams.

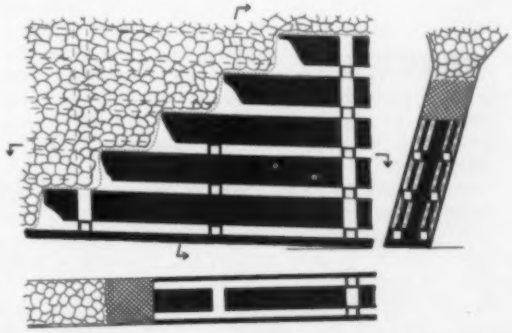

Fig. 5—Underground hydraulic coal mining in the USSR. Steeply pitching seams using flexible shield.

tates maximum coal extraction without contamination with rock. When the bridge collapses the rock above falls and mixes with the coal, resulting in an average loss of 12 pct and a small increase in ash content of the mined coal. The coal is flumed along the sublevels and down the raise to the lower level, where it is crushed and pumped to surface. The monitor is then moved back along the sublevel and mining of another layer of coal is started.

The caved rock may lie at its angle of repose and the coal can be washed down over the rock into the flume. This is possible because rock is heavier and does not move as easily as coal under the influence of the water. Alternatively a flexible shield or mat is used to control the descent of the caved rock and to separate the coal and rock during pillar extraction. The flexible shield consists of four layers of wire netting with a 0.8-in. square mesh and a wire diameter of 0.08 in. The wire netting is laid over the top of the coal in the panel to be mined and descends as mining proceeds, holding back the caved rock from the working area. Where the overburden is thin, the shield is installed from surface. Overburden is removed by dragline and the wire netting is laid on top of the seam and covered by 20 ft of rock by bulldozer. When the shield is installed underground, a horizontal layer of coal is mined at the uppermost sublevel and the wire netting is laid on top of the coal forming the floor.

Operating Results: In the USSR a coal seam 20 to 60 ft thick (average 34 ft) with hanging wall and footwall of average strength and pitching at 65° to 70° to the horizontal is mined by the methods described. In 1953 production per man underground was 235 tons per month. By 1954 it had increased to 385 to 566 tons per month, five to seven times the tonnage obtained at mechanized mines in the same district.

Future Development

The first hydraulic coal mines in the USSR had a planned production of 550 tpd, but as results have been so successful, future mines will be constructed for production of 1650 to 3300 tpd. This is expected to increase further the productivity of hydraulic coal mining.

In New Zealand underground hydraulic mining has produced coal at less cost than any other underground method—in some cases from areas of coal pillars, abandoned because of fire, which could not be mined by other methods. Although it cannot be universally applied, hydraulic coal mining is worth investigating where natural conditions make mechanized mining difficult.

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Cut Exploration Costs with Photogeology

Ore deposits are usually structurally controlled while aerial photos readily provide structural information.

by Kalman N. Isaacs

BY minimizing time that must be spent in the field, intelligent application of photogeology offers tremendous savings in exploration programs. In areas so remote and hazardous that ground exploration must be deferred, photogeology assumes its most important role, often in conjunction with airborne geophysics.

There is still another advantage. Aerial photography may give the geologist a relatively quick understanding of large-scale structures and relationships that are difficult to interpret on the ground.

There are, of course, limitations. Details of structure and stratigraphy are usually out of reach. Only the broadest rock classifications are generally available to the photographer—precise lithologic descriptions must await field checking.

No identification of an orebody can be made from photo study alone. The best the photogeologist can do is to indicate those locations where deposits may possibly occur. In this respect the method is comparable to geophysics.

Ideally, the photogeologist should be a capable and experienced field geologist. This by no means implies that geologists with limited field experience should not try their hand at aerial photography, but for best results with a new technique, broad background experience is desirable. In the specialized field of mining exploration knowledge of structure is particularly important, since ore deposits are usually structurally controlled and aerial photos are more amenable to structural than stratigraphic interpretation.

It is also useful to understand the fundamentals of photogrammetry. This is an extensive and complicated field in itself, but knowledge of the basic quantitative relationships in aerial photos, and the instruments available to the photogrammetrist, is invaluable to the photogeologist. Such information is best set forth in several standard textbooks on photogrammetry[1-4] and on photogeology.[5-7]

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Criteria for Recognition

In studying a feature on a stereoscopic photo pair, the geologist has five general characteristics to consider: 1) topography, 2) drainage pattern and texture, 3) erosion characteristics of the associated soil, 4) vegetation, and 5) tone.

The importance of the gross topography in analysis of a geologic feature is self-evident, but details of the nature of the relief are often equally significant as identifying characteristics. Frequently these details are overlooked.

The nature of the drainage associated with a feature is, again, often diagnostic. Drainage may be described in terms of gross pattern (dendritic, rectangular, trellis) and also texture (fine, medium, coarse). Texture refers to the relative abundance of surface water.

The third factor to be considered is soil erosion. The nature of an immature residual soil is influenced by the rock underlying it and, within broad classifications, soils can be identified on aerial photos by the type of gullies forming in them, as well as by their tone and vegetation. In general, coarse granular soils, such as might develop on a sandstone or a granite, tend to form short gullies with angular, V-shaped cross sections. Clay soils or silty soils, typically associated with shales, form long gullies with rounded, gradual banks and relatively little incision.

Vegetation is occasionally a guide to rock type. A particular shrub or tree may have an affinity for a chemical constituent of a particular rock, and its growth may serve roughly to delineate that rock unit. Conversely, it may have an aversion to the soil overlying a rock unit. In either case, the relative density or scarcity of a type of vegetation, or all vegetation, in an area may have significance. Likewise, permeability and porosity of a rock unit affect the availability of water. This too is indicated by growth of vegetation.

Tone is probably the least important and least reliable of the factors to be considered, as it is usually determined, and strictly limited, by combina-

tions of local conditions—moisture, soil granularity, and type of vegetative cover. Relative tones must usually be considered, since minor variations in photo developing often lend different overall tones to photos within the same area.

(See illustrations pages 468 and 469.)

In considering all these factors, it is necessary to appreciate the scale of the photo. It is the combined aspect of these, together with geological insight, that reveals what the feature is.

Final labeling of a feature as a syncline, fault, or dike must depend on general geologic understanding, as well as detailed knowledge of the characteristics and peculiarities of specific rocks and structures under various conditions of climate and topography.

Research—Maps—Procedure

In performing a photogeologic study, several preliminary steps must be undertaken. Initially, as in all exploration work, an exhaustive library search must be made, and all available geologic and geophysical literature and maps on the area in question must be analyzed and applied to the photo study. Since photogeology is often used as a reconnaissance tool, literature on the area may be scant and highly generalized, or even nonexistent.

In addition to the photos, a base map is necessary, suitable for locating individual photos. Ordinarily this will take the form of an index mosaic, or better yet, a controlled mosaic. If no mosaic or base map is available, the photogeologist must construct one, or at least prepare a chart locating the photo centers as closely as possible.

The purpose of such a map is twofold. It enables the photogeologist to study the photos under his stereoscope systematically and to relate them quickly to the whole area, and it provides in overlay form the base map he uses to plot his conclusions.

After orienting himself until he knows as nearly as possible where each of his photos is located, the geologist may begin his study of the photos.

Next, if a mosaic accompanies the job, it must be searchingly examined. The purpose is to deduce the overall structural pattern and to see if any large-scale units lie within it to form natural subdivisions in the work.

Then the photos are examined, flight line by flight line, and the features recognized are plotted by standard geologic notations and symbols.

Much has been said about the method of annotating photos and recording data. The conclusion usually reached by those who have published on the subject is that a crow quill pen and india ink are preferable because they result in a fine, clean line and hide a minimum of detail. However, pen and ink markings may make it more difficult to use the photos for other purposes. And it is best to be right the first time in marking a feature—ink is permanent.

An accepted procedure is to annotate the photo in grease pencil under the stereoscope and then go directly to a transparent overlay of the mosaic or base map and copy the information down carefully, in detail, with a pencil. A drafted copy of this overlay map can be used with the unmarked mosaic and with the photos, free of markings. Fractures, joints and bedding traces are plotted on the overlay, and even if major contacts are not recognizable, the overall effect of these minor features may make the

Recent vulcanism in French West Africa. A volcano crater is visible in the north central part of the photo. In the southeast a recent lobe of lava has flowed across the northeast-southwest striking metamorphic bedrock. Photo covers about 7500 ft from east to west. (U. S. Air Force)

structure or rock type clear. Attitudes are plotted and dips are defined within relatively broad ranges.

Photogeological Reports

In general, the report submitted by the photogeologist should follow the accepted form for geologic reports. The form may be more or less standard—an introduction, containing geographic and statistical information; a section on stratigraphy and one on structure; a résumé of the geologic history of the area, as much as may be deduced; a section on the economic aspects of the area; a conclusion, summarizing the study; a bibliography; and acknowledgments.

The section on stratigraphy necessarily differs from the usual lithologic description submitted by the field geologist, who describes rocks in terms of albite-anorthite ratios or the degree of roundness in a sand grain. The photogeologist uses terms of relative resistance, influence on drainage, and tone.

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Aerial Photo Interpretation

This photograph of limestone topography in Puerto Rico illustrates the importance of climate in the interpretation of photographic material. Terrain is the tropical equivalent of karst topography. The sinks have developed to such an extent that they have coalesced, forming a *limestone hummock* terrain. In the southern part of the area, the east-west alignment of some of the hummocks suggests some structural control. Photo covers about 3 miles from north to south. (Puerto Rico Dept. of Public Works)

Photogeologic Map

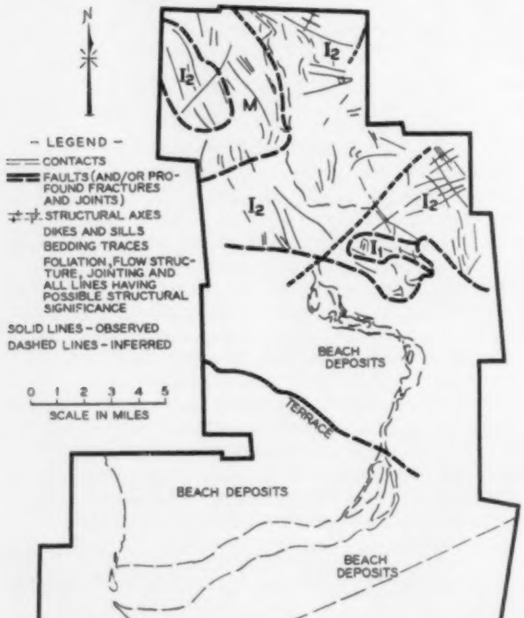

Photo index base map and corresponding photogeologic map. Location is the mouth of the Natashquan River in Quebec, a known area of magnetite and ilmenite beach sands. In addition to the beach deposits, several rock units are distinguishable.

Anticlinal structure in British Columbia. An eroded anticline is seen in southern part of the area. Flanking rocks are those of the Cambrian Caribou series. Rocks exposed in the axial part of the structure are Pre-Cambrian metamorphic. Photograph covers about 8500 ft from east to west. (RCAF)

Major fault is visible in Nova Scotia crossing the photo area from southwest to northeast, passing through the lake in the southwest and embayment in the northeast. West of this line the regional strike is slightly east of north, with a joint system at right angles, and a transverse fault striking northwest-southeast. East of this line, a sharp anticlinal or synclinal fold is evident. The cigar-shaped feature in the east, oriented northwest-southeast, is a drumlin. The photo covers about 9000 ft from east to west. (RCAF)

Tools of the Photogeologist

Ordinarily, adjacent pairs of aerial photos are studied under a stereoscope. Two types of stereoscope are in general use—the simple pocket lens stereoscope and the mirror or prism type.

Pocket Lens Stereoscope: This type is readily portable and therefore ideal for field use. It offers the further advantage of magnification. It does, however, require considerable overlap in the photo pair and thus affords very little scope in the stereo model. When work is done on the overlapped portion, one of the photos must be bent up. Quantitative estimates are deceptive with a lens stereoscope owing to lens distortion, and it is difficult to annotate the photos because the stereoscope supports are in the way.

Mirror Stereoscope: Mirror or prism designs are portable, but too bulky to be used generally as field instruments. They afford a full stereo model and more freedom of movement to annotate the photos, but they do not give a magnified view unless they have binocular attachments. Such binoculars reduce the view and illumination considerably, and they are not as satisfactory for detailed work as the simple lens type. Combined use of the mirror stereoscope for the broad, overall view of a photo pair and the lens stereoscope for the all-important detailed view is the most convenient arrangement. A pair of photos can be viewed stereoscopically with the naked eye, but this involves too much strain for any extensive work.

Parallax Measuring Devices: A second group of instruments in wide use are the parallax devices for measuring relative differences in elevation on a stereo model. These are based on the *floating dot* principle. A pair of dots, one seen with each eye under a stereoscope, is placed over the same point on each of the two photos. The horizontal separation between them is adjusted until the dots merge and float at ground level, and this separation is noted on a gage, usually in hundredths of millimeters. The photo pairs must be oriented precisely along the line of flight in order to obtain accurate results.

For geology the practical use of this type of instrument is in measuring attitudes. Three or more determinations are made on the same geologic horizon and the dip is solved as a three-point problem. In practice, one is lucky to come within 5° of the truth, so as a rule attitudes are recorded on photogeologic maps as low, intermediate, and steep.

Types Available: Parallax measuring devices vary in cost and complexity. The simplest is the parallax bar, or stereometer. Occasionally the large, precision stereoplotting instruments such as the multiplex and the Kelsh plotter have been used to trace beds in areas of low dip, but such machines are normally used in preparing topographic maps.

Economic Relation of Mining Rate to Grade of Ore

by Harry M. Callaway

IN times of a falling metal market the mine operator attempts to remain in competition by lowering his overall cost and raising the grade of ore he processes. But these adjustments can be made only within narrow limits imposed by inherent characteristics of the operation. Is there some additional method whereby the break-even point can be lowered?

One approach, perhaps the only one, to lower a break-even point is careful cost analysis considering both fixed and variable costs in relation to mining rate. The following is an analysis of cost distribution and its significance in determining the most economic combination of the factors of ore grade and tonnage output applicable to a given mining operation.

Perhaps an analysis should begin with a definition of terms, it being understood that such definitions are not universally acceptable, but rather are chosen to facilitate discussion.

Recovery Grade—The percent of a ton of crude ore that, on being processed, reports in the marketable concentrate. Its symbol is R, and it is numerically equal to the inverse of the ratio of concentration multiplied by 100.

Market Price (SP)—Dollar value of a ton of concentrates produced from milling ore.

m—Number of tons mined and milled in one month of operation.

SP x R—Equal to recoverable dollar value of one ton of ore.

Fixed Cost—For a mining operation, that cost which remains constant regardless of number of tons mined. Examples are insurance, depreciation of equipment, capital investment, administrative, management, and accounting costs. Let a symbolize the total fixed cost of an operation for a period of one month.

Variable Cost—Cost that varies directly as the number of tons of ore mined and milled. Examples are blasting powder, a large portion of labor, haulage fuel, bit steel, and hoisting power. Let the symbol b equal the variable cost per ton.

Total Cost—Symbolized by C and equal to the quantity $(a/m + b)$.

Case Studies

Assumptions: The fixed cost per month for a given zinc operation is $40,000. The variable cost per ton of ore mined and milled is $1.70. Milling and hoisting capacity is 24,000 tons per month, and market price for the 60 pct zinc concentrate produced by the mill is $70.00 per ton. Available ore for capacity operation is 6000 tons of 7.5 pct recovery grade, 6000 tons of 6 pct, 6000 tons of 5 pct, and 6000 tons of 3 pct.

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Case 1

Assume the mine operates at one-fourth capacity and that the best available ore is mined, that is, 6000 tons of 7.50 pct recovery grade. Then:

Recoverable value of the ore = 70x0.0750x6,000	=	$31,500
Total cost = $40,000 fixed cost plus (1.70x6,000) variable cost	=	50,200
Therefore, loss for month = $50,200 − $31,500	=	18,700

$$\text{Value per ton} = \frac{31{,}500}{6{,}000} = 5.25$$

$$\text{Cost per ton} = \frac{50{,}200}{6{,}000} = 8.37$$

$$\text{Loss per ton} = \frac{18{,}700}{6{,}000} = 3.12$$

Case 2

Assume the mine operates at one-half capacity and that the best available ore is mined, that is, 12,000 tons of 6.75 pct recovery grade. Then:

Recoverable value of the ore = 70x0.0675x12,000	=	$56,700
Total cost = $40,000 fixed cost plus (1.70x12,000)	=	60,400
Therefore, loss for month = 60,400 − 56,700	=	3,700

$$\text{Value per ton} = \frac{56{,}700}{12{,}000} = 4.72$$

$$\text{Cost per ton} = \frac{60{,}400}{12{,}000} = 5.03$$

$$\text{Loss per ton} = \frac{3{,}700}{12{,}000} = 0.31$$

Case 3

Assume the mine operates at three-fourths capacity and that the best available ore is mined, that is, 18,000 tons of 6.16 pct recovery grade. Then:

Recoverable value of the ore = 70x0.0616x18,000	=	$77,616
Total cost = 40,000 plus (1.70x18,000)	=	70,600
Therefore, profit for month	=	7,016
Value per ton	=	4.31
Cost per ton	=	3.92
Profit per ton	=	0.39

Case 4

Assume the mine operates at full capacity and that in addition to the 18,000 tons of 6.16 pct mined in Case 3, the remaining 6000 tons of 3.00 pct available ore are mined and processed. The total would be 24,000 tons of 5.37 pct recovery grade. Then:

Recoverable value of the ore = 70x0.0537x24,000	=	$90,300
Total cost of month's operation = 40,000 plus (1.70x24,000)	=	80,800
Therefore, profit for month	=	9,500
Value per ton	=	3.76
Cost per ton	=	3.36
Profit per ton	=	0.40

Discussion: Note that recoverable dollar value per ton of the additional 6000 tons in Case 4 is only $2.10 (market price of concentrates multiplied by recovery grade), whereas cost per ton for mining the entire 24,000 tons of available ore is $3.36. Does this mean that mining the additional tonnage of low grade material caused a loss of $1.26 per ton? Profit from the operation was highest in Case 4. Therefore,

the operation extracted profits from a material that was submarginal in grade. How can this apparent contradiction be reconciled.

In Case 3 the cost is \$3.92 per ton, compared with \$3.36 in Case 4. Therefore, mining the additional 6000 tons of low grade ore lowered the cost of the previously mined 18,000 tons by \$0.56 per ton, a total saving of \$10,080. Cost of mining the additional 6000 tons is \$20,160 and its recoverable dollar value is \$12,600. Therefore, a loss of \$7560 must be charged against a savings of \$10,080, giving a balance of \$2520 over and above the profit realized on the 18,000 tons mined under Case 3.

It is often economical, therefore, to mine submarginal grades of ore, thereby redistributing fixed cost over larger tonnages and reducing the average cost per ton.

Obviously there is a cut-off grade below which a loss, rather than a profit, will accrue. What is this lower limit?

The cut-off grade should fall at a point where the dollar value of the *last* ton of rock mined is exactly equal to the cost of mining that ton. This does *not* mean that the dollar value of each additional ton must equal the *average* cost per ton. Once enough of the better grade ore has been mined to pay for the entire *fixed cost* of the operation, then each additional ton of ore must have only the dollar value necessary to pay for the cost of mining the additional ton. By definition, this value is the variable cost per ton.

The Break-Even Formula

Derivation: To break even, total cost of production must equal exactly the total market value of the product. For purposes of analysis, it is possible to examine the relation of unit cost to unit value of the product without destroying the validity of the break-even relationship. Thus, to break even on the mining and milling of one ton of ore, the cost must equal exactly the dollar value of the concentrate extracted from that ton.[1] (Refer to symbols previously defined.) Then, to break even,

$$SP \times R = C = a/m + b$$

or, solving for R,

$$R = \frac{a/m + b}{SP}$$

This expression relates recovery grade to market price of concentrates and to cost of production at the break-even point. Its usefulness is infinitely increased by incorporation of the factor m, thus relating mining rate to distribution of costs. The accompanying graph shows the curve of this formula, plotted for values discussed under Cases 1 through 4.

Rate of Change: In times of falling metal prices it is often contended that it is economical to mine the better grade ore from a given deposit at a sacrifice of tonnage output.

The following relation is derived from the fundamental break-even expression:

$$SP = \frac{C}{R}$$

First inspection of this relationship may lead to the hasty conclusion that an increase in the value of R will decrease the value of SP. That is, if the recovered grade is increased, the mining operation will show a profit under a proportionally lower market price for concentrates. This is the thesis of the *high-grader*. It is valid, but valid *only* if the increase in R does not cause a corresponding and compensating increase in C.

The anti-thesis recognizes that cost varies inversely as the tonnage and is a minimum at the point where production is a maximum, thus concluding that any increase in R gained by reduction in tonnage will cause cost to rise at a prohibitive rate.

The real answer lies between these extreme positions at a point that is peculiar to the individual operation. Obviously, a mine that has unlimited high grade ore blocks can increase grade without reducing tonnage. Other operations may find a drastic cutback in production is necessary to secure even a modest increase in grade. The proper solution, therefore, will be obtained by a quantitative estimate of the *rate of change* of grade with respect to tonnage in the break-even expression.

The break-even formula and the graph calculated from it express recovery grade as a function of tonnage output:

$$R = f(m) = \frac{a/m + b}{SP}$$

where a, b, and SP are constant at a given time for a given operation.

$$R + \Delta R = \frac{\dfrac{a}{m + \Delta m} + b}{SP}$$

$$\Delta R = \frac{\dfrac{a}{m + \Delta m} + b}{SP} - \frac{\dfrac{a}{m} + b}{SP}$$

$$\Delta R = \frac{\dfrac{a}{m + \Delta m} - \dfrac{a}{m}}{SP}$$

$$\Delta R = \frac{a \cdot m - a(m + \Delta m)}{m(m + \Delta m)} \times \frac{1}{SP}$$

$$\Delta R = \frac{a \cdot m - a \cdot m - a \cdot \Delta m}{SP(m^2 + m \cdot \Delta m)}$$

$$\frac{\Delta R}{\Delta m} = - \frac{a}{SP(m^2 + m \cdot \Delta m)}$$

$$\frac{dR}{dm} = \lim_{\Delta m \to 0} \frac{\Delta R}{\Delta m} = - \frac{a}{SP \cdot m^2}$$

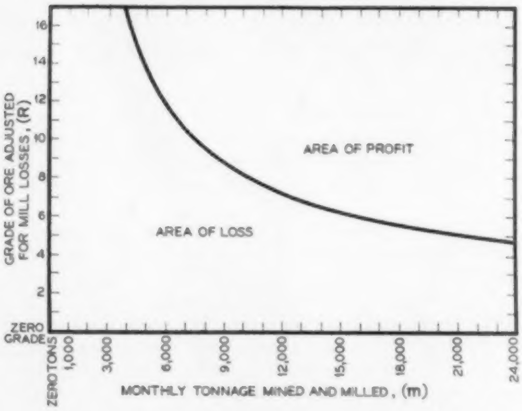
Therefore, the rate of change of grade with respect to tonnage at any break-even position is

$$- \frac{a}{SP \cdot m^2}$$

For the operation discussed under the case studies the rate of change at the 20,000-ton position is as follows:

$$\text{incremental } R = - \frac{a}{SP \cdot m^2}$$
$$= \frac{\$40,000}{\$70 \times (20,000)^2}$$
$$= 1.43 \times 10^{-6}$$

The solution is in terms of decimal values of grade per ton. A more useful and meaningful unit



The Break-Even Curve.

of expression would be in terms of percent change in grade for 1000-ton change in production. Therefore, multiply the above answer by 100 (decimal to percent) and by 1000 (ton to 1000 tons). Incremental $R = 1.43 \times 10^{-6} \times 10^{5} = 0.143$ pct per 1000 tons.

This means that at a production level of 20,000 tons, a cut-back in production of 1000 tons must be accompanied by an increase in recovery grade of 0.143 pct to compensate for the increase in cost due to the lower production level. It should be noted that the rate so derived carries the notion of an *instantaneous* rate of change and is the rate only at the beginning of the interval from $m = 20{,}000$ to $m = 19{,}000$. Over large intervals an integration could be used to establish the needed compensating grade. However, the most direct method is by two calculations from the break-even expression. Suppose it is desirable to find the minimum increase in grade necessary to offset increased cost due to a cut-back in production from 24,000 to 15,000 tons per month. The break-even grade, R, is calculated for the values $m = 15{,}000$ and $m = 24{,}000$.

$$R = \frac{\frac{a}{m} + b}{SP}$$

$$R = \frac{\frac{40{,}000}{15{,}000} + 1.70}{70}$$

$$R = 6.24 \text{ pct recovery grade}$$

$$R_1 = \frac{\frac{40{,}000}{24{,}000} + 1.70}{70}$$

$$R_1 = 4.81 \text{ pct recovery grade}$$

$$R - R_1 = 1.43 \text{ pct}$$

The differential, $(R - R_1)$, is the needed increase in grade.

Conclusions

1) The point of maximum profit coincides with capacity production. This statement is true even when, to achieve capacity production, it is necessary to mine submarginal grades of ore.

2) The cut-off grade below which it is not economical to mine is that grade which has a recoverable dollar value exactly equal to the variable cost per ton. It is the redistribution of fixed cost over large tonnages that lowers the cost per ton and allows the cut-off grade to approach its lower limit.

3) The most economic combination of grade and tonnage factors occurs at a point that is peculiar to the individual operation and is determined by examination of the rate of change of grade with respect to tonnage in the break-even expression.

Reference

[1] H. M. Callaway: Basic Break-Even Formulas Devised to Simplify Mine Evaluation. *Engineering and Mining Journal*, November 1954, pp. 90-92.

Discussion

A reviewer's comment submitted on H. M. Callaway' article is published here with the author's reply:

Comment: This seems over-simplified. In the first place, the *fixed cost* item is too all-inclusive, as tonnage drops from 24,000 to 6000 in the illustrations and should also be a variable. Secondly, the variable cost, in my opinion, does not vary directly as the number of tons mined and milled. Therefore, to apply an exact solution, such as calculus, to assumptions that in themselves are debatable seems to me to leave room for considerable doubt. In our own operations variable and fixed costs are not as exact as the author indicates. For instance, the fixed costs on capacity operations are not the same as fixed costs at 25 pct of capacity.

H. M. Callaway (*author's reply*): In effect, the reviewer states that fixed cost becomes at least partly variable and variable cost becomes somewhat fixed as tonnage increases. By definition, such polymorphous behavior is ruled out. However, in an operation where mill capacity, hoisting capacity, haulage equipment, size of staff, etc., are not in their balanced relation to each other, it is quite likely that additional inputs of fixed cost may be necessary at definite positions of production. An example would be the salary of an additional shift boss or distributive purchase price of an additional truck or drill needed to secure additional breaking and haulage in excess of a certain production level. In such instances the relation of grade to mining rate remains valid. However, the curve plotted for such an unbalanced operation, though having no change in slope, *would* have points of discontinuity at positions of tonnage where the additional input unit of fixed cost occurs. The gap in grade would be calculated by use of the formula for the curve using both the lower and higher total fixed cost figures.

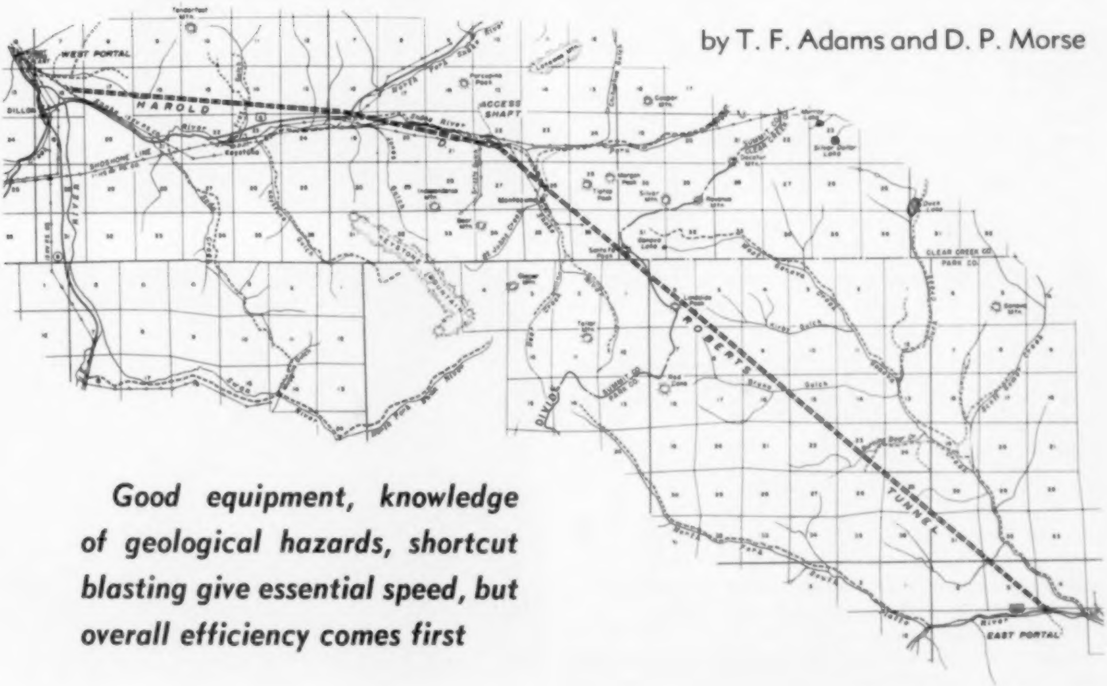
Another reason for the critic's comment may have been the present writer's failure to elaborate sufficiently on the definitions. It should not be interpreted that the writer failed to recognize that certain items have components of both fixed and variable cost. The components, however, can be broken out (to a practical limit) and properly classified as if they were separate cost items rather than segments of a single cost.

In answer to the remaining remark that "variable cost does not vary directly as the number of tons mined," the author must confess he does not understand. It is suspected that "variable cost" in the operation cited fails to vary directly as the tonnage mined to the same degree that someone has failed to properly classify his costs.

Organized Speed —

Key to Successful Tunnel Results

by T. F. Adams and D. P. Morse



Good equipment, knowledge of geological hazards, shortcut blasting give essential speed, but overall efficiency comes first

Tunneling is primarily an excavating cycle consisting of a sequence of operations: drilling, shooting, ventilating, mucking, and erecting supports, if necessary. However, the type and condition of the ground that is being worked, as well as the size and shape of the tunnel, determine the rate of advance. In addition developments in drilling, ventilation, mucking, and haulage equipment have contributed to faster tunneling.

Geology: This discussion will be limited primarily to rock tunnels or those which require the use of explosives, and the cost and hazards depend primarily on the condition of the rock which may be broken, decomposed, and contain water with gas. Advance geological study is an important part of any tunnel work and helps to minimize serious miscalculations. The geology itself, at best, is a matter of experience and prediction and it is not so much a matter of the types of rock the tunnel may go through, but a knowledge of the class and intensity of the rock defects that may be encountered.

In limestone underground channels and solution caverns may contain heavy flows of water or a mixture of sand and water. In sandstone, such as the Dakota in Colorado, continuous flows of water may be encountered, especially if the beds intersect a river or stream. Water is not normally a problem in shale, but considerable water may be encountered wherever the material is overlain by a water-bearing formation. In shale, especially if coal seams are present, methane may be found. If anhydrite is present, water may change it to gypsum and cause swelling; calcium sulfate may also develop and attack the concrete lining. Deadly hydrogen sulfide may be present, as well.

In schists and gneisses, if altered chemically, squeezing and swelling conditions may be encountered. Although these rocks are normally not too hazardous, inflows of water may be encountered and overbreak that must be filled with concrete if the tunnel is lined can become a serious problem. In granites, diorites, and monzonites conditions are usually good, but in traversing mineralized areas or fault zones they may be very difficult. Water and hydrothermal alteration through fractures or faults can completely upset original cost estimates, as they cannot necessarily be predicted from the surface. In volcanics the rock may be completely decomposed or unconsolidated and water and gases may be present in large quantities.

Geology has a decisive influence on the difficulties and costs of tunnel construction and even moderate knowledge enables the builder to predict and anticipate hazards up to a point.

T. F. ADAMS and D. P. MORSE are, respectively, Project Manager and Superintendent, Blue River Constructors, Dillon, Colo.

Rate of Advance: Tunnel driving methods depend on rock behavior and size and shape of the tunnel cross section. Popular methods include full face, heading and bench, and pilot tunnel. Numerous other methods, are used, dependent upon conditions, but for simplification the full face method will be considered here, as it is now the one most commonly used.

Pneumatic rock drills, drill jumbos, drill jibs, carbide bits, mechanical mucking devices, better haulers, and improved ventilation systems have all had a part in speeding tunneling operations. The use of steel supports instead of timber saves erection time and reduces tunnel width.

A rate of advance estimate is the controlling factor in all tunnel contracts and is determined both by conditions and size of the tunnel. Size controls the number of drills that can be used and this in turn controls the drilling time as well as the type of loading and hauling equipment that can be used.

Excavating Cycle: As a specific example we will refer to the $45-million Harold D. Roberts tunnel currently being driven for the City and County of Denver, Board of Water Commissioners. The tunnel will be driven with four headings and extend over about 23 miles to divert water from the western slope to the eastern slope of the Continental Divide. Considerable water and bad ground that required grouting and close support have been major problems at the west portal, near Dillon, Colo. At the east portal, near Grant, Colo., progress is as expected even though the ground has required more support than anticipated. From a shaft near Montezuma, Colo., two headings are to be driven—one west to Dillon and the other east to Grant. The building contract is held by Blue River Constructors, a joint venture of six contracting companies.

In this tunnel operation the drill jumbo is brought into the heading, platforms extended, and drills placed in position by hydraulic jibs. This fills the whole tunnel so it is necessary to complete all mucking from the previous round before any drilling can be started. The drilling pattern used is a standard pyramid cut—the number of holes varying between 35 and 50 according to the type of rock. On an average, about 7.5 ft of hole is pulled and drilling will vary from 3 ft in bad ground to 11 ft in hard ground. The average drilling cycle takes about 1 hr with four machines on the jumbo. Close supervision is needed, as spotting of the holes may result in a poor round or too much overbreak.

Powder used is 1¼ x 8 or 1¼ x 16-in. and loading a round takes about 20 min. The powder has low density with a good degree of water resistance and cohesiveness and has a bulk strength of about 45 pct. The fumes are excellent and this type of powder detonates with a uniform velocity which gives uniform control. The holes are loaded to within 1 to 1½ ft of the collar. No stemming is used. While not the most efficient way to use powder, speed is the essence of the cycle.

A few minutes before blasting, the blower is reversed to exhaust from the heading. This allows the crew to return to the heading within a short time. Electric blasting caps are used with delays running from 1 to 10. Primer is loaded at the bottom of the hole and the holes are hooked up in parallel. The jumbo is removed from the heading and the round detonated with 220 v from a switch at least 1000 ft from the heading.

Drill, Shoot! Muck Out

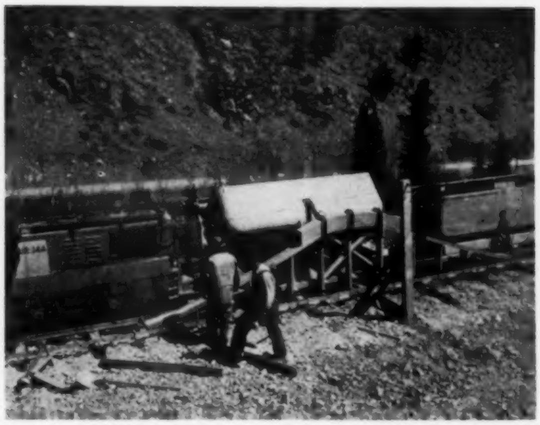
After shooting the mucker is brought in and loading out of the material started. Short sections of track are laid to advance the mucker and bring the jumbo to the heading. These sections are later replaced with the standard length of rail. By the use of a california switch—a double track that slides on the main single line track—the empty cars are shunted to the heading singly and loaded cars are removed. Mucking usually takes about 1½ hr, including delays.

Upon completion of mucking, necessary supports are placed in line and grade and blocked with wood blocking. Distance between the steel supports varies with the ground, but, as a general rule, 4 ft are used for moderate rock load, 2 to 3 ft for heavy rock, and 5 ft for light loading. If, however, the rock requires support at the face, the spacing should be an even fraction of the length of shot pulled.

Installation of utilities such as fan line, air line, water line, pump lines, power cable, lights, transformers, pumps, and track is carried on behind the heading while the normal cycle is taking place. Sidings are also laid at regular intervals to store equipment and supply cars.

Equipment: Wet drilling is a must and portable jumbos, mounting four or more drills on jibs, are used to expedite drilling and to serve as platforms for loading, connecting up the round, and setting

Dump, Set Supports, Recycle



supports. The platforms are made to fold up against the frame of the jumbo when not in use for clearance to pass other equipment. Space is provided on them for drills so that time is saved in moving the drilling equipment in and out of the heading. In this particular case, 3½-in. machines with 5-ft shells are used mounted on hydraulic jibs that can be controlled in two directions. Hollow carbon 1¼-in. drill steel and 1⅝-in. carbide bits are used. Tunnel service is severe on bit life and obtaining the proper type is a vital cost element.

Three mucking machines per heading are employed—one in use, one for standby, and one in repair. The cars are 5-cu yd, or about 7-ton capacity, and about 15 cars are filled per round. A train normally consists of one 8-ton diesel locomotive with scrubber and seven cars. Size of the 24-in. gage, 56-lb track used was determined by the size of the tunnel as the equipment is normally not more than twice as wide as the track gage. Positive blowers are used for ventilation—all located at the portals. Compressed air is supplied through an 8-in. line and about 2700 cu ft of air is required per heading. Water is supplied through a 4-in. line under 80-lb pressure. Power is supplied through a three-conductor cable carrying 4160 v with necessary transformers for the lighting and power circuits. On the downhill headings pump lines must be carried with the necessary pumps and an arbitrary figure of 10,000 gpm was used for capacity on this job. This is based on a sudden inflow of that amount of water. However, by the use of feeler holes ahead of the face and grouting when necessary, it is expected that this amount of water can be kept to a minimum.

Organization: For this size and type of tunnel the overall crew per heading will include about 105 men. There will be about 25 men per shift underground or a total of 75 men for three shifts. About 18 men are required on the surface for outside work, and supervision will require about 12 men.

In this particular case the drill sharpening shop is located at the west portal and supplies steel for the entire contract. The main office is also located at the west portal and in addition a yard is maintained on the outskirts of Denver for the purpose of storing material and supplies. All purchasing is handled through the Denver yard in order to centralize and expedite the procurement of materials.

Labor relations are an extremely important phase of tunnel work. Since this type of work is based on cooperation and efficiency as well as repetition, it is essential that the crews develop a sense of competition—not only between crews on various shifts, but between headings. This is what makes a tunnel crew, and, of course, helps set footage records. Footage, however, is not the criterion for driving a tunnel. If overbreak is neglected in getting footage, it is quite possible to lose whatever advantage may be obtained on driving a tunnel by having to concrete the excess overbreak. Overall efficiency is the primary consideration.

Future Developments: The ideal tunnel would be one which had been driven from end to end without necessitating a change in the originally selected working methods. However, the difficulty in altering tunnel routines after the job has started is that the working areas such as headings are poor places in which to experiment in getting out rock. Of course, if original plans do not prove applicable, or, if conditions warrant alteration of heretofore successful procedures, no time should be lost in correcting conditions.

Recent developments have indicated that the burn cut with large center holes has proven to be successful in uniform rock. Roof bolting in uniform material is also taking hold. There have been several developments on a continuous mucking cycle which eliminates changing cars.

There is a great need for a type of drill capable of far faster drilling. More efficient loading equipment is also needed and considerable work needs to be done on ventilation problems.

Radical changes in hard rock tunnel driving do not seem too apparent at the moment and it is mainly a business of gradual modifications or improvements. A few minutes cut off a certain phase of the cycle is essentially the limit. In soft rock a continuous rotary miner has been developed and is in use. If, through the use of a rotary-type oil well bit, the same thing could be done, hard rock could be overcome. This is being considered and has been used in small shafts and raises. There are numerous possibilities, but unfortunately the average contractor has to get rock out of the heading and has neither the time nor the finances to experiment beyond a certain point.

Hydraulic Backfilling

- ***Effects on Mining Methods***
- ***Use of Sands and Slurries***
- ***Trends in Current Practices***
- ***Possible Future Developments***

by Richard Maclin Stewart

A priest at Shenandoah, Pa., one day persuaded the president of the Philadelphia & Reading Coal & Iron Co. to slush breaker waste and culm into old mine workings in order to save his church from being destroyed by surface subsidence. The year was 1864, the project was successful, and history recorded the first use of hydraulic backfilling. Shortly afterward, the method was applied to several eastern coal mines for area fill to control subsidence. In 1884, hydraulic backfilling was used to control a mine fire in the Schuylkill region of Pennsylvania.

The idea was taken to Europe by German engineers and developed successfully at the Myslovitz colliery in Upper Silesia in 1901. From there it was applied to several mines throughout Europe.

The practice of area filling was taken to South Africa in 1909 to the Village Gold mine in the Transvaal.

At the start of World War I mill tailings were being used for mine backfill on a limited scale in the Cripple Creek district of Colorado. By the end of the war the method was introduced by the Anaconda Co. in Butte, Mont., to fight mine fires. Since 1917 more than 7½ million tons of mill tailings have been placed hydraulically in the Butte mines.

During the 1920's, many world mining districts used fill material placed as a slurry. The Mataham-

R. M. STEWART is Assistant to the Director of Mining Research, The Anaconda Co., Butte, Mont.

Survey of

	Screen Analysis			
	Mesh	Wt, Pct	Mineral Composition	Pulp Density, Pct
American Smelting & Refining Co. Galena Mine Wallace, Idaho	+35 +48 +65 +100 +150 +200 +325 −325	0.80 2.80 7.20 14.80 17.40 20.20 19.80 17.00	Siderite, quartzite, some pyrite	68 to 74
Calera Mining Co. Cobalt, Idaho	+65 +100 +200 +325 −325	3.0 10.0 25.0 22.0 40.0	Biotite, schist, phyllite, quartzite, minor pyrite and pyrrhotite	54 to 60
Day Mines Inc. Wallace, Idaho	+65 +100 +150 +200 −200	5.0 18.0 20.0 17.0 40.0	Quartz, quartzite, siderite, ankerite, sericite, chlorite, minor pyrite	68 to 71
Anaconda Co. Butte, Montana	+35 +48 +65 +100 +200 +400 −400	5.7 15.0 30.6 24.3 18.3 4.7 1.4	Quartz, feldspar, sericite, minor pyrite	65 to 70
Homestake Mining Co. Lead, S. D.	+80 +100 +150 +200 −200	2.8 10.1 20.7 28.0 38.4	Cummingtonite, biotite, quartz, 7 to 8 pct sulfides	50 to 65
Kerr-Addison Gold Mines Virginiatown, Ont.	+65 +100 +150 +200 −200 +325 −325 −10μ	0.2 0.8 6.2 24.1 31.7 37.0 1.0	Silicates and carbonates of Ca, Mg, Fe	55 to 60

* For purposes of comparison to other hydraulic fill installations

A fast, inexpensive means of providing permanent ground support in an active stope.

Plant Operating Conditions and Costs

Dry Solids, Sp Gr	Percolation Rate, In. Per Hr	Pipeline Size, I.D.	Friction Loss	Maximum Pulp Transportation Distance, Ft: Horizontal	Maximum Pulp Transportation Distance, Ft: Vertical	Pumps	Filling Costs	Per Ton Ore, $	Per Ton Fill, $
3.2	2 to 3.06	3-in. steel pipe, some 2½-in. rubber lined	No data	1780	2380	None (gravity system)	Labor:		
							Sand plant operators	0.06	0.13
							Mechanics and electricians	0.02	0.04
							Stopers (preparation and pour)	0.09	0.17
							Mechanics (lines and support)	0.05	0.10
							Electrical		
							Communication	0.01	0.02
							Maintenance	0.02	0.03
							Supplies	0.06	0.11
							Compressed air	0.02	0.03
							Power	0.02	0.03
							Miscellaneous	0.02	0.03
							Total	0.37	0.69
3.0	No data	4-in. wood stave and 4-in steel pipe	14.2 ft of head per 100 ft in nominal 4-in. pipe at 400 gpm, 58 pct solids	2800	230	None (gravity system)	Dewatering and desliming	0.006	0.012
							Plant operation, maintenance, and power	0.169	0.360
							Pipeline	0.023	0.050
							Bulkheads	0.153	0.330
							Clean-up	0.014	0.031
							Total	0.365	0.783
2.9	4 to 8	3-in. nominal steel pipe	No data	2500	750	None (gravity system)	Total	0.67	1.23
							Anselmo Filling Costs*	**Per Ton Ore**	**Per Ton Fill**
2.9	4 to 8	4-in. nominal steel pipe (rubber lined 3½-in.) 5-in. wood stave pipe on surface	4.9 psi per 100 ft in 5-in. wood stave pipe at 70 pct solids, 310 gpm, 5.07 ft per sec	1100 ft (2-in. pipeline) 4000 ft (3-in. pipeline)	250 ft 3800 ft	Lexington 40-hp A. S. H. C Frame 1170 ft horizontal in 5-in. wood stave pipe	Water	0.012	0.023
							Electricity	0.008	0.016
							Plant operation	0.115	0.231
							Shaft pipe	0.001	0.001
							Level pipe	0.006	0.013
							Filter walls	0.059	0.119
							Fill labor	0.090	0.180
							Clean-up	0.029	0.058
							Fringe benefits	0.034	0.068
							Total	0.354	0.709
							Filling Costs		
3.0	2	6-in. steel pipe (rubber lined 5½-in.), surface to 2600 level 5-in. steel (rubber lined 4½-in.), 2600 to 4100 level 4-in. steel pipe in stopes	No data	2460	745	None (gravity system)	Dewatering plant operation and maintenance		
							Pipeline maintenance		
							Bulkheads, drainage		
							Clean-up		
							Actual running of sand (desliming and storage ore metallurgical charges.)		
							Total	$0.18 per ton ore	$0.36 per ton sand
							Filling Costs, 1956	**C&F Stope, $**	**Open Stope, $**
2.88	3.5	6-in. pipe (rubber-lined 5 9/16-in. Flexible hoses on each level, 5 9/16-in. Extra heavy black pipe on levels 3 53/64-in.	Approximately 21 ft per 100 ft in nominal 4-in. level lines at 60 pct solids, 86 dry tons per hr, total volume 352 gpm	1480	316	None (gravity system)	Total tons mined	922,233.00	
							Total cubic yards placed	380,314.00	318,170.00
							Total cost, $	171,296.39	149,764.07
							Costs per cubic yard of fill, $		
							Desliming plant, operation and maintenance	0.14	0.12
							Pipeline per cubic yard	0.02	0.02
							Spread in stope	0.18	0.08
							Bulkheads, miscellaneous (clean-up)	0.11	0.15
									0.37
							Total	0.45	0.37

railroad loading and freight costs are not included.

bre mine in Cuba was one of the first to make extensive use of hydraulic backfilling as an integral part of their mining method. They also pioneered in the use of rubber lined pipe as a transporting medium.

Homestake Mining Company was one of the first in the U. S. to exploit simultaneous hydraulic backfilling as a part of the mining cycle. In 1932 they developed a method of handling their washed cyanide tailings in pipelines and introduced the sand portion for stope fill.

After World War II many mines in the west and in Canada applied hydraulic backfilling methods and developed mining cycles based on the use of classified mill tailings. This progress was due in part to the increased experience within the minerals industry in the handling of solids in liquid suspensions. The perfection of rubber-lined slurry pumps and the expanded technology of materials handling in pipelines, together with the application of wet cyclones for dewatering and desliming, led to economical applications in the mining field.

Fill Sand Sources

Of the several possible sources for hydraulic fill sands, the simplest is mill tailings, when mine and mill are integrated. Alluvial sand or gravel deposits can also be considered as a source of fill material when the mill is located at a distance from the mine or when mill tailings are unsuitable.

Physical Requirements

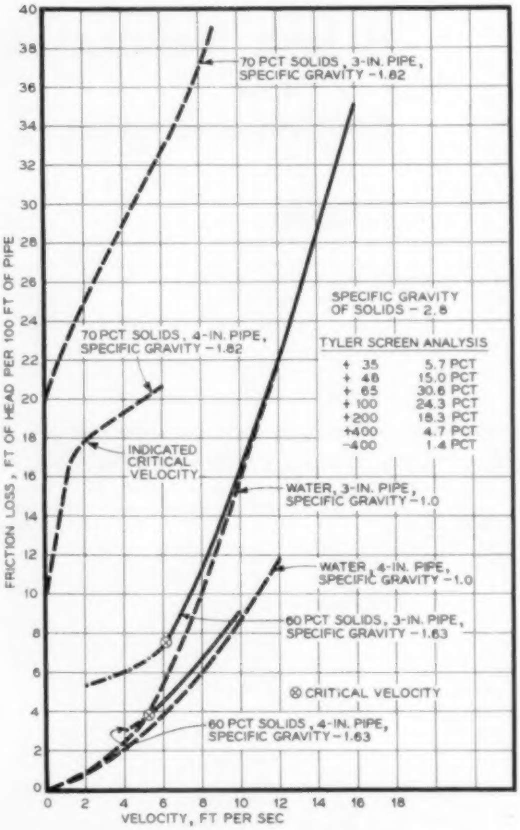
Hydraulic fill material must have the following physical properties:

The sand material must be chemically inert. Specifically, cyanide must be carefully controlled in order to avoid contamination of air circulated through working places. Sulfur content must also be controlled to avoid excessive heating of the fill by oxidation of sulfides and to prevent emanation of sulfur dioxide gas. A maximum of about 8 pct pyrite or 4 pct pyrrhotite can be tolerated and is beneficial in that the oxidation results in cementing the sand particles.

In some cases Portland cement is added to the slurry in small amounts to promote cementation. This is also used when valuable minerals occur as fine particles and it is desirable to prevent dilution with sand fill. A weak concrete slab is made by adding Portland cement to the fill slurry in the surface plant, and topping off the fill pour with this cement mixture. Anaconda's experiments in Butte showed that ore particles penetrated a maximum of 8 in. when blasting on fresh sand fill with no additives.

Particle Size and Percolation Rate: When fill is part of the standard stoping cycle, it is essential to use a sand product that dewaters rapidly. This allows the miners to safely enter the stope immediately after filling and also permits blasting immediately after a pour. The problem is to maintain fluid characteristics while transporting the sand and then to quickly stabilize the sand when it is in place to avoid the hazards of extreme hydraulic heads.

Research work has shown that it is the extremely fine portions of a fill sand that determine its dewatering characteristics. The quantity of -20μ (-800 mesh) material determines permeability. Through elutriation and infrasizer tests Anaconda found that a maximum 3.5 pct by weight of -20μ particles could be tolerated.



Friction loss of Anaconda tailings in standard steel pipe.

A simple test for permeability was developed in Canada. Percolation rate, in inches of water penetration per hour, gives an excellent field control. In this test the slurry is run into an 1⅜-in. diam tube to a depth of 12 in. The sand settles and water is added at the top of the tube. A filter cloth at the bottom of the tube allows the water to drain through the sand. When the filtration rate becomes constant (usually about 30 min) the percolation rate is recorded. Most mines have standardized on a water percolation rate of at least 4 in. per hr. Experience has indicated that when the rate is less than this, there is apt to be a dewatering problem underground.

It is desirable to have a minimum of voids in the consolidated sand fill so that the sand will resist ground movements and effectively stabilize the mine opening. A wide distribution of particle sizes is required to assure many points of contact between particles in order to distribute the wall rock forces into the fill material without displacement. The coarsest rock sent underground in pipe lines is $-\frac{3}{4}$ in., but particles of this size are mixed with fine sand and are a small fraction of the solids by weight.

Specific Gravity: The specific gravity of solids used for fill varies considerably and is proportional to the pulp velocity needed to keep the sand particles in suspension in the pipeline. The specific gravity of the minerals ranges from 2.4 to 3.6.

Pulp Density or Percent Solids by Weight: The particles are transported as a sand-water slurry—in one operation moving as a continuous flow from

surface to the stope underground. Fill can be poured into place at the rate of 100 tph of solids through a 4-in. pipeline, thus filling at the rate of 2000 cu ft per hr. It is this simple rapid means of placing hydraulic fill that makes the method more economical than conventional tramming methods. But it is the solids that are really wanted in the stope, not the water, so most plants operate at relatively high pulp densities. They range from 60 to 74 pct solids, by weight. Critical pulp density varies slightly depending on the specific gravity of the minerals.

In one plant using tails with a specific gravity of 2.9, the critical pulp density is 74 pct solids. That is to say, below 74 pct solids the pulp mixture will flow like a liquid. There is enough water to fill the voids between particles plus an excess of water which prevents contact between particles. When the pulp is agitated the particles stay in suspension and plastic flow occurs. However, at 74 pct solids and above plugging conditions occur in the pipelines even at high velocities. On a volume basis, it has been demonstrated that when over 50.5 pct of the pulp is water, the mixture will flow if solids are kept in suspension by adequate pipeline velocities.

Flow Characteristics: There is relatively little information to be found on the flow characteristics of thick slurries in pipelines, and it presents a fruitful field for research. In the adjoining graph concerning Anaconda fill sand, friction loss, in feet of slurry per 100 ft of pipe, is plotted against slurry velocity. The data indicate that critical velocity, i.e., the minimum velocity at which sand grains will stay in suspension, is 6 fps at 60 pct solids and about 2 fps at 70 pct solids. The data also indicate that, at pipeline velocities of more than 12 fps, pulp of 60 pct or less solids shows friction losses approaching the values for water.

If we consider a pipeline velocity of 6 fps, the following friction head losses per 100 ft occur in 3-in. pipes: water, 5.4 ft; slurry of 60 pct solids, 7.3 ft; slurry of 70 pct solids, 32.8 ft. In a 4-in. pipe, the values are: water, 4.0 ft; slurry of 60 pct solids, 4.5 ft; slurry of 70 pct solids, 20.6 ft. The flow of Anaconda hydraulic fill becomes plastic above 60 pct solids and approaches a solid plug flow at 74 pct solids.

Operating conditions and costs at several hydraulic fill plants are shown on pages 476 and 477.

Types of Plants

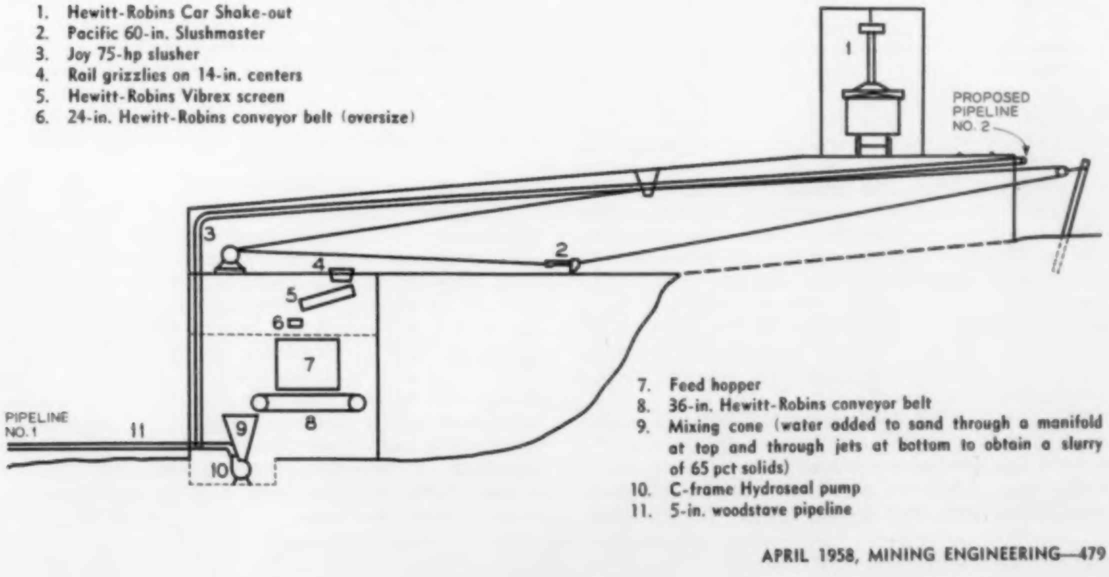
Fill Source Remote from the Mine: The type of surface installation is determined by the source of sand fill. At Butte, Mont., and at International Nickel Co. in Canada, dewatered mill tailings are transported to the mine in railroad cars. This damp product is either unloaded with the help of vibrators to storage bins or, as at some Butte plants, it is sluiced from the cars with high pressure water nozzles and the pulp is collected in agitation tanks. This method depends on hydraulic monitors to break down the damp tailing and resulting pulp density cannot be positively controlled. Density will vary between 50 pct and 60 pct solids unless the damp sand freezes in storage, in which case the pulp density may be as low as 30 pct solids by weight.

In order to control pulp density when utilizing a damp tailing product, Anaconda designed and built the Lexington hydraulic fill plant at Butte. There, a measured amount of damp sand is fed to a mixing cone where a measured amount of water is added to make a slurry with desired pulp density. The plant normally operates in the density range between 65 pct and 70 pct solids. Let us consider pulp of 70 pct solids for a moment. This pulp would by definition contain 30 pct moisture by weight. We have found that fills will retain 10 pct moisture for several years. Thus drain water from the stope is 20 pct of the pulp by weight. When our plant is operating at the rate of 100 tph of dry sand we must drain water at the rate of 7600 gal per hr. However, if the proportions of sand and water were not controlled in mixing and pulp was sent underground with 50 pct solids, the water would drain from the stope at the rate of 21,300 gal per hr. Thus the dewatering problem can be reduced from 21,300 gal to only 7600 gal per hr by increasing the pulp density from 50 pct solids to 70 pct solids.

Reducing the amount of drain water has many advantages. It means less pumping, fewer leaks in

Lexington Hydraulic Sand Fill Plant

1. Hewitt-Robins Car Shake-out
2. Pacific 60-in. Slushmaster
3. Joy 75-hp slusher
4. Rail grizzlies on 14-in. centers
5. Hewitt-Robins Vibrex screen
6. 24-in. Hewitt-Robins conveyor belt (oversize)



7. Feed hopper
8. 36-in. Hewitt-Robins conveyor belt
9. Mixing cone (water added to sand through a manifold at top and through jets at bottom to obtain a slurry of 65 pct solids)
10. C-frame Hydroseal pump
11. 5-in. woodstave pipeline

the stopes, less segregation of the fill material, less slimes carried through the filter bulkhead (reducing cleanup), lower pipeline velocity (lessening wear), and finally, faster draining and better consolidation.

Integrated Mine-Mill Plant: The integrated plant generally employs dewatering and desliming facilities in the mill. Three types are used: The Matahambre type, with bowl classifier, yields a pulp with densities of 50 to 60 pct solids. In the second type, mill tailings are fed into a storage tank and the slimes and water are reduced by decantation. The third type employs wet cyclones for dewatering and desliming. The cyclone underflow is an ideal product for simultaneous stope back filling. Outstanding small plants of this type can be seen at Knob Hill mines in Republic, Wash., and at the Galena mine, near Wallace, Idaho.

An outstanding large cyclone plant operated by Kerr-Addison Gold Mines Ltd. at Virginiatown, Ont., produces 2000 tons of deslimed tailings per day from a very fine mill tailing averaging 91 pct −200 mesh and 30 pct -10μ. A two-stage wet cyclone operation reduces slime content to only 1 pct -10μ and produces a pulp with a percolation rate of 3.5 to 4 in. per hr. Kerr-Addison has two storage tanks for mine fill, each 35 ft diam, 63 ft high, with capacity of 4000 tons of sand.

Mining Methods Utilizing Hydraulic Fill

Several mining methods have been developed to utilize hydraulic back fill. Basically, there are two advantages: 1) A faster filling cycle, hence a faster mining cycle which produces more tons per unit time and dollar invested. 2) Better ground conditions, which usually permit use of a less expensive mining method. At Mt. Con mine in Butte (traditionally a square set mine with heavy ground), hydraulic fill improved the ground support so much, compared to waste rock fill, that timber was eliminated. Rock bolts are now used to support any bad ground. The saving in material and labor reduced the relative stoping cost 30 pct and increased efficiency by 50 pct to 13 tons per man shift.

Hydraulic fill also improves mine ventilation by confining air flow to the desired air courses. Old workings are effectively sealed and are therefore fire proof.

Horizontal Cut and Fill: The horizontal cut and hydraulic fill method has been developed since World War II with the help of advances in mining equipment. At Butte, 30-hp electric slusher hoists combined with 42 or 48-in. scrapers have proved most economical for slushing ore on the sand fill. Anaconda tried an Eimco 630 with 32-cu ft hopper, and showed that vehicles can be used on fresh stope fill. Although the cost of the first trial was equal to the standard hoist and scraper system, the full-tracked loader proved more economical than the scraper during the operating phase. The difficulty was in lifting this large, heavy piece of equipment in preparation for a fill. In Europe, Copco tire-mounted overshot loaders with hopper body have been developed for use on sand fill in Sardinia. In the future, we may see conventional earthmoving equipment with low pressure tires adapted to use underground on sand fill.

The use of a self-loading transporter on sand fill necessitates relatively close chute spacing so the vehicle can dump its ore load with a minimum of travel time. The steel pipe chutes at Day Mines fit admirably into the stoping cycle—prefabricated steel segments are quickly assembled to the height of the next pour, usually 6 to 8 ft. In Canada, at Kerr-Addison and International Nickel, concrete chutes are used. Prefabricated forms are easily placed and quicksetting cement is poured through the hydraulic fill pipeline into the chute forms. The circular or rectangular chutes are 4 to 5 ft across and the concrete wall is about 12 in. thick. Many mines also use some type of timber cribbing in manways and chutes, but the material used is dependent on the tonnage. In the future, as more mechanical equipment is applied to underground metal mining, steel and concrete chutes will probably be more widely used due to ease of placement and long life.

The only stoping methods integrating hydraulic filling into the operating cycle are variations of the horizontal cut and fill method. In Butte, when mining veins from 3 to 20 ft wide, it is possible to expose an old fill in the endline of an adjacent stope after allowing only 4 to 6 weeks for consolidation of that fill. Of course, in the cut and fill method the old fill is exposed for only a 10 to 12 ft height and very little dilution occurs.

There have been cases where it was necessary to mine hanging wall vein splits after a hydraulic fill stope had been completed on the foot wall. In some instances the old fill actually formed the foot wall of the new stope and in other cases there was a *horse* between the old fill and the new stope.

Panel Stopes: With good wall rock, it is possible to delay the filling operation until stoping has been completed. Then a block type fill is poured after sealing all downchutes with concrete bulkheads. Block fills are often used with the following mining methods: shrinkage, sublevel, open stoping, and room and pillar. The objective of the hydraulic fill is to stabilize a large block of ground and prevent ground movement and the possibility of rock bursts and air blasts. After consolidation and cementation, pillars or stopes may be mined adjacent to the fill.

To reduce stope development work, several mines have developed modifications of shrinkage stoping methods in which a stoped-out block is filled to a predetermined elevation. Prefabricated forms are then positioned on the fill and concrete is poured to establish ore handling facilities for the block above. Finger raises and scram drifts are constructed on the old fill. When the concrete has set, the forms are removed and hydraulic fill is poured around the structure. Stoping operations are then started in the new block. Often the rock is dropped through existing ore passes which were carried up in the fill of the old block. In this manner rock handling facilities are constructed in old stopes prior to filling with a consequent saving in development work.

Summary

Hydraulic backfilling presents a fast, inexpensive means of providing permanent ground support in an active stope and it may prove a valuable tool in the design of stoping methods for the future. It permits opening up larger areas since they are open for only a fraction of the normal time and ground movement can be controlled with positive support. Rock burst and subsidence can be prevented. Hydraulically placed sand fill provides a horizontal surface on which scrapers, railroads, track-mounted equipment, and wheeled vehicles can be effectively operated. It therefore makes possible the use of larger, faster, heavier, rock moving equipment in underground operations.

Collected Discussion of

Energy-Size Reduction Relationships In Comminution

(MINING ENGINEERING, page 80, January 1957, AIME Trans., vol. 208)

by R. J. Charles

Discussers: F. C. Bond, D. W. Fuerstenau, J. A. Holmes, A. J. Lynch

F. C. Bond: This is an outstanding paper on comminution theory and represents a considerable advance in mathematical formulation. It clears the way for a discussion that should ultimately decide whether the work index should be considered as a parameter according to the Third Theory or as a constant.

In the empirical differential equation

$$dE = -C\,dx/x^n \qquad [1]$$

either C or n must be treated as a variable to satisfy known crushing and grinding conditions. The author has chosen the exponent n as the variable, with C as a constant.

According to the Third Theory, in the general case n is a constant equal to 3/2, and C is a variable related to the work index Wi. If x is placed equal to P (the size in microns which 80 pct of the product passes) and F is the size 80 pct of the feed passes, then C equals 5 Wi, and

$$\int_0^W dE = \int_F^P -5\,Wi\,dx/x^{1.5} =$$

$$W = \frac{10\,Wi}{\sqrt{P}} - \frac{10\,Wi}{\sqrt{F}}$$

which is the basic equation of the Third Theory.

It is apparent that this integration involves the error of treating the parameter Wi as a constant, just as the author's treatment involves the error of integrating his variable n as a constant. These errors are probably relatively unimportant, considering the small range of the variables. When more is known about the factors contained in the work index, Eq. 1 can be revised as a partial differential equation.

The Third Theory exponent of 1/2, or $n = 3/2$ in the Charles' differential equation (Eq. 1), was not chosen primarily because it is the average between the Kick and Rittinger exponents; this may be merely fortuitous, since it is illogical to define truth as the average between two errors. It was first discovered from the study of a large mass of crushing and grinding data and confirmed by laboratory testing. The general correspondence of the Third Theory equation to actual crushing and grinding results over the entire size reduction range argues strongly that a fundamental phenomenon is involved.

The *a posteriori* theoretical explanation of the constant Third Theory exponent is that the work done in breaking rock is proportional to the length of the crack tips formed. Compressive force applied to the rock results in deformation and strain energy. When this locally exceeds the breaking strength a crack tip forms, the surrounding strain energy flows to the crack tip and extends it to split the rock, with release of the energy as heat. Under rapid impact the crack tip may form before the strain has reached equilibrium, resulting in less energy expenditure and probably a coarser product. In any case, the elastic limit of a brittle material is practically the same as its breaking strength, and the work necessary to form the crack tip is the work required to break.

The crack length of a ton of broken rock cannot be measured directly. However, the particle shapes of feed and product are similar, and the crack length is considered equal to the square root of one half of the new surface area formed. It is, therefore, proportional to $1/\sqrt{D}$, where D is the equivalent product particle diameter. This is the basis of the Third Theory equation.

Many secondary factors can be imposed upon the crack length breakage. These may cause the Third Theory work index to increase or decrease somewhat as the product size becomes finer. The work index is properly a parameter rather than a constant, making it an increasingly valuable practical criterion of the work input required in crushing and grinding.

The Transactions papers appearing in MINING ENGINEERING during 1958 will subsequently appear in AIME Transactions Volume 212, and may be permanently referenced to that volume.

The product size distribution is an important function of the work index, but does not completely control it. Many factors affect the work index by changing the product size distribution and otherwise. The existence of flaws or incipient cracks, bedding planes, natural grain sizes, and crystals are some structural factors affecting the work index. Dynamic factors include: amount and velocity of impact breakage, circulating loads, dilution, ball and rod sizes, mill speeds, etc.

Any analysis of the many factors affecting comminution becomes exceedingly complex and can only be done piecemeal. The point to be made here is that the Third Theory work index is the most nearly constant quantity available, as well as the most readily obtainable, and in the writer's opinion the most acceptable theoretically; therefore, the analysis of any factor is most accurately made by treating it as a function of the work index. Use of the variable exponent proposed by Charles would abandon the first approximation already available and would permit much larger errors.

The author's treatment is based on the assumption that the Schuhmann size distribution slope $\underline{\propto}$ remains constant as the product size decreases. Experience has shown that when the natural fines are not removed the slope normally decreases with the product size. When 80 pct of an average material passes 1 in. or more the slope approaches unity, where 80 pct passes 48 mesh it is approximately one half, and when 80 pct passes 325 mesh it is 0.4.

Charles selected the Schuhmann power law size distribution rather than the Rosin-Rammler exponential type or the log-probability plot. None are entirely satisfactory, although theoretical considerations seem to favor the exponential law.

He offers four experimental tests of the variability of the exponent n. Each of these tests involves a special case, with feed particles all of one size or closely sized between the three screen sizes from ⅜ in. to 6 mesh. According to the Third Theory n remains constant for the general case when the feed contains particles of all sizes in their natural weight distribution down to the grind limit.

It is probable that the value of n increases as the fines are artificially removed from the feed. When the fines are removed n also increases as the reduction ratio decreases. This explains the tests confirming the Rittinger theory in which a closely sized feed was used. The Third Theory equation requires an empirical correction in the feed size parameter when fines are removed from the feed.

The work index can continue essentially constant for different materials independently of whether the characteristic Schuhmann slope $\underline{\propto}$ is large or small. This is illustrated in the laboratory ball mill grindability test results tabulated beow:

Ore Copper	Anaconda		Utah	
Specific Gravity	3.22		2.86	
Grind. Test No.	1477A		938A	
Circulating Load, Pct	250		250	
Mesh Size Tested	**∝**	**Wi**	**∝**	**Wi**
28	0.715	12.2	0.615	12.8
35	0.710	12.0	0.610	10.8
48	0.695	10.9	0.610	10.5
65	0.680	11.0	—	10.6
100	—	12.5	—	10.6
150	—	12.4	—	10.2
200	—	—	—	11.7
Average	0.700	11.71	0.612	11.03
Standard deviation, pct	—	6.32	—	8.82

There can be no doubt that the work index criterion is more practical.

The Third Theory equation permits the rapid calculation of required work input when only one size parameter is known—the size 80 pct passes. Variations in size distribution slope are automatically reflected in the Wi value, and complete screen analyses are not required. On the other hand, use of the exponent n as the sole variable would require size distribution information that is often not available and would greatly restrict work input calculations.

The evidence appears to favor Charles' choice of n as a variable to correct for artificial adjustment of the feed size distribution, usually by screening out fines. When a natural feed is used, n appears to remain constant at 3/2, and the work index Wi varies, both with the material being tested and the machine operation. Standard laboratory tests define Wi for any particular material. Changes in machines, impact velocities, circulating loads, and other operating factors cause changes in both the product 80 pct passing size and the size distribution parameters, resulting in work index variations and consequent changes in mechanical efficiency. Much experimental work remains to be done before these factors can be evaluated.

R. J. Charles *(author's reply)*—The author appreciates F. C. Bond's interest in the work under discussion and would like to comment on his several points.

The Third Theory equation, written in its general form, is as follows:

$$W = \frac{10W_i}{\sqrt{P}} - \frac{10W_i}{\sqrt{F}} \text{ where } W_i = f(F,P) \qquad [19]$$

The equation can be generalized no further, since the function f, which defines the parameter W_i, can only be determined experimentally for each specific case to which the whole equation is applied. Thus, in general, when all boundary conditions are set the term W_i varies with product and feed size. In this writer's opinion an equation of the type

$$W = K\left[\frac{1}{k_P{}^{\gamma}} - \frac{1}{k_F{}^{\gamma}}\right] \qquad [20]$$

where k_P and k_F are size moduli solves the above difficulty of a varying constant, for when the constants K and γ are chosen correctly there is no necessity of bringing in a parameter to adjust the equation to fit experimental evidence. As a consequence, since it is a constant, it may be integrated and is not subject to the criticism in paragraph four of the discussion. Similarly, on the above basis, this writer objects to the statement: "The Third Theory work index is the most nearly constant quantity available, as well as the most readily obtainable."

The derivation of the general energy-size reduction equation was not predicated on any specific type of size distribution. The Schuhmann power law distribution was used only as an example that is apparently applicable to a number of experimental cases. Further, this writer is not aware of any theoretical considerations that favor any specific type of size distribution for a ground product.

With respect to the tabulated data in the above discussion the W_i values appear to be relatively constant. As a consequence the n value for both sets of data would be approximately 1.5. No stipulations would be placed on the corresponding a values except that they be greater than 0.5, which is apparently the case.

It is difficult to draw comparisons of practicality of something that has been in use for a relatively extended period of time and something that has not been tested. The Third Theory has been used with great success for many industrial problems and in many cases this writer advocates its continued use. He feels, however, that it is an inadequate mathematical description of the physical process of size reduction and as a consequence is likely to hinder further advancements of the scientific aspects of the problem.

D. W. Fuerstenau: The article by Charles has put considerable light on some of the problems of comminution, yet one important factor in his article appears to have been neglected. The question arises as to how the theory given can account for the effect of changes

of feed size in a comminution process on the energy required for a specific size reduction. In the derivation of the specific energy-size reduction relationship, used to correlate experimental results, the influence of feed size has been neglected by an approximation. Also in the experimental results all tests were performed from a constant and usually sized feed material. Any theory that can satisfactorily describe the comminution process must account for both the initial and final sizes of the particles involved, and for the work in question the reasons for dropping the terms that account for feed size in the derived equation must be explored more deeply.

R. J. Charles *(author's reply)*—Fuerstenau brings up an important point that still requires experimental investigation. The approximation referred to is, of course, only valid if the size reduction is large. If this is not the case then the feed size term should re-enter the general equation even though a relatively complicated procedure must then be followed to solve for the necessary constants. For the most general case, where the feed and product size distributions are similar, the general energy-size reduction equation leads, as indicated in the article, to the following:

$$E = \frac{C\,\alpha}{(n-1)\,(\alpha-n+1)} \left[k_p^{1-n} - k_f^{1-n} \right] \qquad [21]$$

where k_p, k_f = size moduli of product and feed; C, α, n = constants; and E = energy.

The above equation may state the case correctly; however, it is possible that a large change in feed size may alter operating conditions and therefore the equation could not be expected to hold. Unexpectedly, the above equation provides a key for testing whether or not the feed size term can be neglected. If the term $(\alpha - n + 1)$ appears negative then the approximation is not valid.

J. A. Holmes: The importance of the excellent article by Charles in stimulating a new and more flexible attitude towards size reduction problems cannot be over-emphasized. The tendency in the past has been to accept one or another of the dogmatic principles with which studies of this subject have been plagued and to pay insufficient attention to the really important characteristics of a crushing or grinding operation, namely, the properties of the material undergoing fracture and the conditions under which stresses are applied. The author of the article has indicated the true academic nature of the so-called laws of comminution and has shown that they have no direct application to crushing and grinding problems in which the quantity of useful product obtained per unit of energy input to a more or less inefficient machine is the important criterion of operation.

It is evident from Charles' results and from those obtained by the writer[18] that for any one grinding machine the value of n is determined by the properties of the material and by the variation of the efficiency of the machine in treating different sizes of particle. However, there remain these questions—what effects do different material properties and different conditions of reduction have on the efficiency of different machines, and how are these effects reflected in the general equation developed independently by Charles and by the writer. In this respect Charles' consideration of the following comments would be of interest.

It is claimed in the article that for quartz ground in a ball mill the same value of n was maintained down to very fine sizes, the finest products being assessed by measurement of surface areas. One would perhaps expect some change in n to occur as grinding progressed due to a change in the efficiency characteristic of the ball mill as the material became very fine. Since no such change was measured it would be interesting to know more about the conditions of grinding and the fineness of grind achieved in these tests. It is well known that assessments of ground products by surface area measurements are critically dependent on the method of measurement used, so that Charles' opinion on the krypton gas adsorption method in relation to grinding problems of this sort would also be of interest.

For the tests on the rod mill and ball mill it was assumed that the energy input was proportional to the grinding time. It would be interesting to know whether this assumption was supported by measurements over the whole range of reduction achieved, since it seems possible that, as the material became finer, changes in the coefficients of friction between the grinding surfaces and in the viscosity of the pulp might result in changes in the configuration of the ball or rod charge.

Charles finds that for pyrex, quartz, fluorite, and cement rock the expression $\alpha - n + 1$ tends to zero. Other materials showing well developed cleavage did not obey this rule, and since distributions of these materials deviate most markedly from the Schuhmann equation, so that α can have little significance, these exceptions are not unexpected. However, tests on quartz performed by the writer[18] in a dry grinding laboratory ball mill gave values for n and α of 1.77 and 1.07, respectively, as compared with 1.88 and 0.91 obtained by Charles in a wet grinding mill. For the dry grinding test $\alpha - n + 1 = 0.30$, as compared with 0.03 for wet grinding. Relatively large values of this expression were obtained for other inhomogeneous ores tested by the writer, even though the size distributions followed the Schuhmann law reasonably closely. It would seem, therefore, that the value of $\alpha - n + 1$ is not necessarily close to zero but is dependent on the properties of the material and on the conditions of reduction. This seems a more acceptable conclusion for two further reasons: 1) the Schuhmann law is, at best, only an approximation of the true size distribution, so that any fundamental relation between α and n seems unlikely, 2) because as $\alpha - n + 1 \to 0$, Charles' constant $A \to \infty$.

If this conclusion is correct it follows that Charles' comments on Kick's law require revision, for as n tends to the value 1 no implicit limitation is then imposed on the value of α. However, the writer agrees that the tendency is for α to decrease as n decreases and that Kick's law may be applicable for reduction of material of such fineness that its resistance to reduction becomes independent of particle size.

It is clear from Eq. 15 that the elegant graphical method of determination of n cannot be used when x_m is not large compared with k. Since this is often the case in large-scale milling practice, where stage grinding through relatively small reduction ratios may be employed, it would be of value to know how Charles would treat such a problem. Also it is not normal grinding practice to feed particles of one size, the case for which Eq 15 was derived. The writer has recently formulated a similar equation:

$$E = K \left[\frac{1}{P^r} - \frac{1}{F^r} \right] \qquad [22]$$

where r = the Kick's law deviation exponent, $= n - 1$. This equation refers to reduction of one Schuhmann distribution with an 80 pct passing size F, to a similar distribution with an 80 pct passing size P and differs from Eq. 15 in that the factors of $\frac{1}{Fr}$ and $\frac{1}{Pr}$ are the same. Charles refers briefly in his article to the treatment of unsized feeds, but in view of the above difference his more detailed comments would be of interest.

Reference has been made to results obtained in dry grinding tests on quartz in a laboratory ball mill. These results are compared in Table II with those obtained from Bond's tests under the same conditions and with those reported in the present discussion.

Although more precise details of the structural properties of the quartz and of the conditions of reduction are necessary for a complete understanding of the observed variations in n and α, these figures serve to illustrate the importance of these properties and con-

Table II. Comparison of Values of n and α for Quartz

Author	Machine	Material	Location	α	n
Charles	Rod mill, 10-in. diam (wet)	Quartz	Not given	0.91	1.90
Charles	Ball mill, 10-in. diam (wet)	Quartz	Not given	0.91	1.86
Hukki	Frictionless crusher	Quartz	Not given	0.93	1.88
Bond[17]	Ball mill, 12-in. diam (dry)	Quartz	California	—	1.79
Holmes[16]	Ball mill, 7½-ft diam (wet)	Quartz	California	—	1.70*
Holmes[16]	Ball mill, 12-ft diam (dry)	Quartz	Cornwall, U. K.	1.07	1.77
Holmes[16]	Ball mill, 7½-ft diam (wet)	Quartz	Cornwall, U. K.	—	1.68*

* Calculated figure.

ditions in determining the behavior of any one material.

R. J. Charles *(author's reply)*—Holmes' comments are especially appreciated in view of the fact that, simultaneously with this work, he has presented similar arguments, reasoned from a more fundamental basis, for the variable exponent concept of energy-size reduction. The interested reader is urged to study Holmes' work,[16] for in this writer's opinion it contains a complete solution for a long-standing problem in comminution.

With reference to the comment on fine grinding of a variety of quartz the experimental results show a possible trend (see Fig. 10) to increased resistance at the finest sizes, ($k_p \approx 5\mu$). Gas adsorption surface area measurements, while reproducible to 2 or 3 pct, are generally considered to give values within 10 pct of absolute and thus the above slight deviation from linearity, if real, could not be clearly delineated. Continuous torque measurements were not made during the investigation; however, particular attention was paid to mill design in that lifter bars were installed to aid in maintaining constant ball and rod configuration. Wattmeter readings, which give only rough indications of power actually consumed by small mills, remained essentially constant during the tests. On the assumption that the size distributions in these tests remained similar it must be concluded that the efficiency characteristics of the mill remained relatively constant.

Since the variable exponent n is limited by the value of α and since experiments conducted by both Holmes and this writer on a number of materials show that there is a marked tendency for n to decrease as α decreases, the possibility that n and α are related should not be discounted. The relationship may be only a consequence of the assumptions used in describing size distributions, but in any event it is worthy of study. The constant A does not approach infinity because the term $(\alpha - n + 1)$ approaches zero but because the term $(n - 1)$ approaches zero. In the statement regarding Kick's law the term $(\alpha - n + 1)$ was considered small, positive, and finite.

If the feed size is not large compared to the product size then the approximation referred to is, of course, not valid and the feed size term must re-enter the equations. For the case of similar feed and product size distributions integration of the general energy-size reduction equation leads to the same equation cited by Holmes. For different feed and product size distributions a slightly different equation is obtained. In any case, experimental determination of the constants required is still most easily made by testing in the range where product size is far removed from feed size. Once n is determined the feed size term can be introduced for a complete solution.

A. J. Lynch: This method of determining energy-size reduction relationships in comminution is a refinement of the general method of such determinations which has been used in various forms by Rittinger,[2] Kick,[3] Bond and Jen Tung Wang,[18] and Bond.[7] As such it is open to the same errors as these others, especially where heterogeneous ores, as defined by Gaudin,[12] are concerned.

The article is based on the Schuhmann equation for size distribution and is applicable only to solids which, after comminution, have a sizing analysis that may be defined by that equation. However, it was pointed out by Roller that this equation rarely applied to more than the finest 15 pct by weight of the particles and that in many cases the size distribution is not, in fact, a straight line but a smooth curve.

It is obvious from an inspection of Figs. 2, 4, 5, 8, and 10 that the actual product sizing analysis may differ markedly from the interpolated sizing analysis and the actual 100 pct passing size may vary greatly from the interpolated value. However, no attempt has been made to account for these variations in the calculated relationship.

The grindability is regarded as the power required for a given reduction of the solid under standardized conditions or the reduction of the solid achieved by the expenditure of a given amount of power under standardized conditions. In effect the constant n, which is a measure of the energy required to reduce the solid from some interpolated 100 pct passing size to another, is a direct function of the grindability in the same manner that the work index used in the Third Theory is a direct function of it.

The theory proposed by Charles is, in essence, similar to the Third Theory except that the criterion of size is an interpolated 100 pct passing size instead of an actual 80 pct passing size. However, the possible errors inherent in it appear to be smaller than those in the Third Theory because the measure of grindability used takes into account the full size range and is less influenced by abnormal conditions at any particular size range and is therefore more accurate.

R. J. Charles *(author's reply)*—The author appreciates the comments by A. J. Lynch and is pleased to offer the following reply. The general energy-size reduction equation, given in the article, is not based on any specific size distribution and the Schuhmann equation was used only as an illustration of how the general equation may be applied. A complex size distribution makes integration of the general equation difficult, but in principle it can be done. To avoid carrying out individual graphical integrations for specific cases, accuracy is sacrificed, for the sake of generalization, by use of an approximation. Further, it is the author's opinion that even a serious deviation from the Schuhmann equation can be tolerated in the specific procedure outlined, since all that is required is that the theoretical size modulus k be a measure of the lateral shift of similar size distribution plots.

The constant n is not analogous to the work index used in the Third Theory, for the work index is a proportionality parameter similar to the generalized constant A in the present work. The main difference between the Third Theory and the present work is that a variable exponent, dependent on material and means of processing, is advocated rather than a fixed exponent.

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Seismic-Refraction Method in Ground-Water Exploration

by William E. Bonini and Eugene A. Hickok

IN the course of an investigation directed toward expanding ground-water facilities in Essex and Morris counties, New Jersey, the Board of Water Commissioners of the city of East Orange authorized a seismic-refraction survey[1] for the purpose of delineating bedrock topography below unconsolidated overburden. Results of the survey were highly satisfactory and led to the preparation of a comparatively detailed bedrock contour map. Knowledge of the bedrock depth and configuration was an important aid in selection of sites for test drilling.

The portion of the East Orange Water Reserve under consideration is in the flood plain of the Passaic River about 10 miles west of Newark, N. J. The flood plain is about 175 ft above mean sea level and is bordered by low hills rising to elevations of approximately 250 ft. The bedrock underlying the Water Reserve consists of sandstone and shale of the Triassic Brunswick formation and is covered everywhere by deposits of unconsolidated glacial outwash sand and gravel, lacustrine clay, and recent river silt as much as 150 ft thick. Yield of wells in the sandstone and shale averages 100 to 200 gpm. Since production wells constructed in the sand and gravel aquifer in the buried river valley shown on the contour map (Fig. 1) yield 300 to 1400 gpm, it was proposed to locate additional production wells in this buried valley, where the yields per well would be maximum.

In 1939 and 1946 the East Orange Water Dept. had electrical-resistivity surveys made to determine depths to bedrock. From the resistivity data the exploration company prepared a bedrock contour map. A well field expansion program begun in 1955 utilized this information to locate sites for test wells along a predicted northward extension of the buried valley in which existing production wells are located. After several test wells (wells 201-205) had been drilled, it became apparent that the resistivity information was unreliable.* For example, test well 201 recorded bedrock at a depth of 72 ft, whereas the resistivity depth determination was 130 ft. As a consequence, the test drilling program was temporarily suspended and a seismic survey was undertaken to determine the topography and extent of the buried valley known from well records to underlie the existing well field.

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In the first phase of this study, several seismic shot point locations were placed at sites where well logs had been obtained previously. This procedure is necessary in a new area to determine whether the seismic method is applicable and what degree of accuracy is to be expected. At the East Orange Water Reserve, depths obtained from the shot points near test wells 202, 203, and 204 were within 8 to 11 pct of the depths logged (Table I). With this assurance that accurate results could be obtained, additional seismic spreads were located on the Water Reserve.

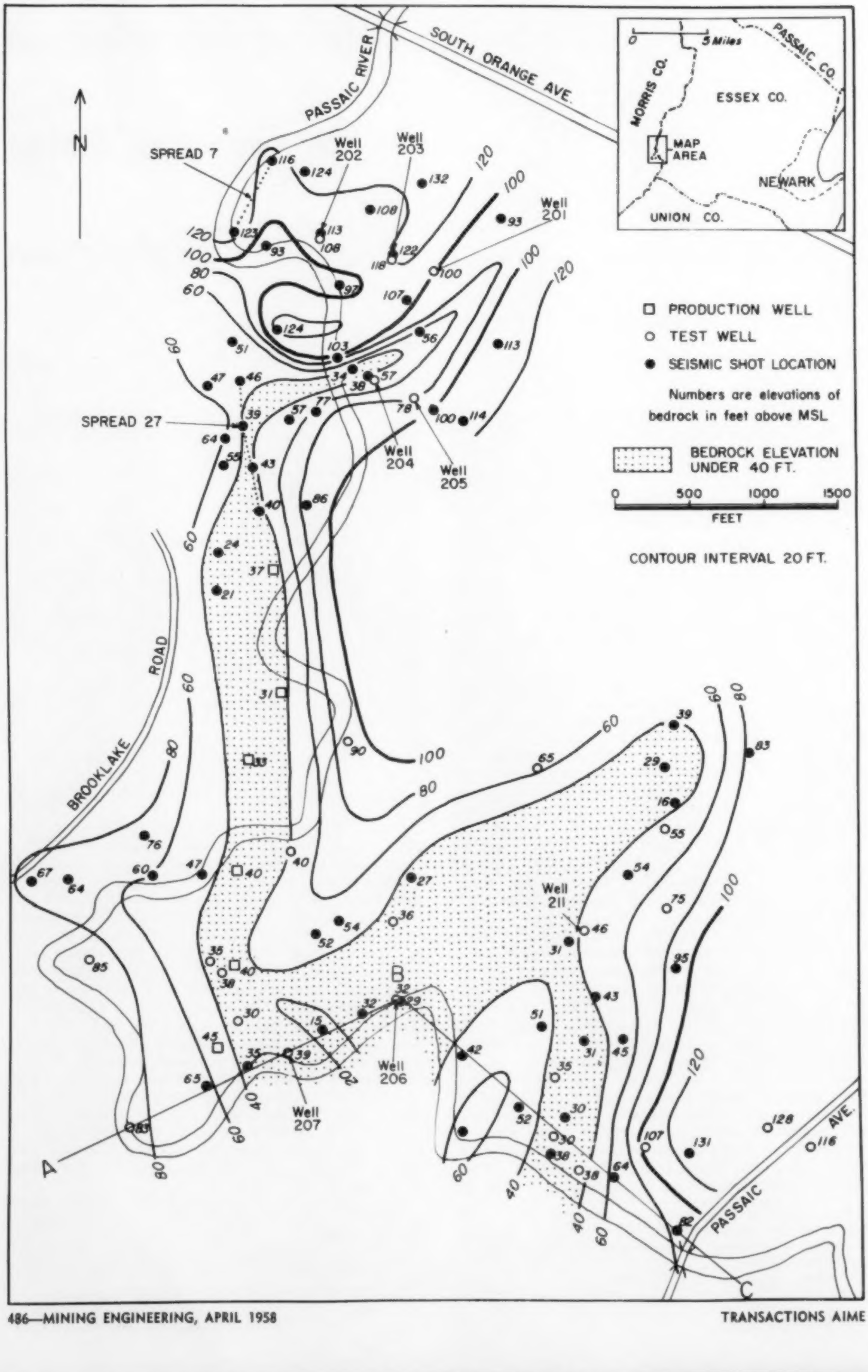
Using a portable refraction seismograph, in the fall of 1955 a crew of four men shot a total of 29 reversed seismic spreads in a period equivalent to six field days. Charges as heavy as 3 lb of 40 pct dynamite were necessary at a few places to overcome ground vibrations caused by traffic on nearby highways. At most other sites, a 1-lb charge was sufficient.

Travel-time plots were made for all spreads, and depths and true velocities were calculated according to formulas for multiple sloping layers by Ewing, Woollard, and Vine.[2] The plot of spread 7 (Fig. 2) is typical of the short spreads in which bedrock was shallow—about 50 ft in this case. Where there were not enough arrivals through the bedrock to define the high velocity bedrock line, the spreads were lengthened. This was done by placing shots on line several hundred feet away from each end of the line of geophones. It was then possible to construct complete reverse plots for both short and extended shot points (see spread 27, Fig. 3). Four individual depths were calculated from each extended spread.

Three and in some cases four seismic layers were observed. The surficial layer had a velocity range of 900 to 1200 fps, the lowest velocity recorded. This seismic layer is above the water table and is interpreted as recent river silt. The bedrock had the highest velocities, which ranged from 10,600 to 16,400 fps.

Intermediate velocities ranged from 4500 to 6800 fps. In every case the intermediate layer was within

* Details of the resistivity surveys were not available to the writers, so it was not possible to analyze these data to determine why the method, successful in other areas, gave poor results in this case.


SOUTH ORANGE AVE.
PASSAIC RIVER
N
SPREAD 7
Well 202
Well 203
Well 201
SPREAD 27
Well 204
Well 205
BROOKLAKE ROAD
Well 211
Well 206
Well 207
PASSAIC
AVE.
A
B
C
0
5 Miles
PASSAIC CO.
MORRIS CO.
ESSEX CO.
MAP AREA
NEWARK
UNION CO.
PRODUCTION WELL
TEST WELL
SEISMIC SHOT LOCATION
Numbers are elevations of bedrock in feet above MSL
BEDROCK ELEVATION UNDER 40 FT.
0 500 1000 1500
FEET
CONTOUR INTERVAL 20 FT.

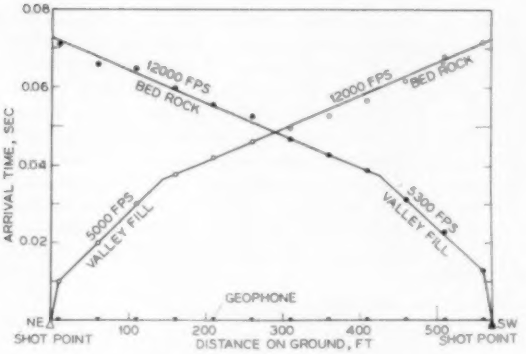


Fig. 2—Travel-time plot for seismic spread 7.

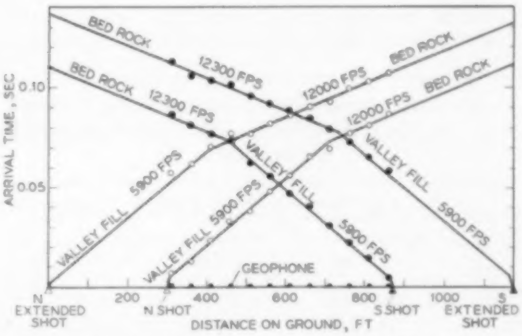


Fig. 3—Travel-time plot for seismic spread 27 with extended shot data.

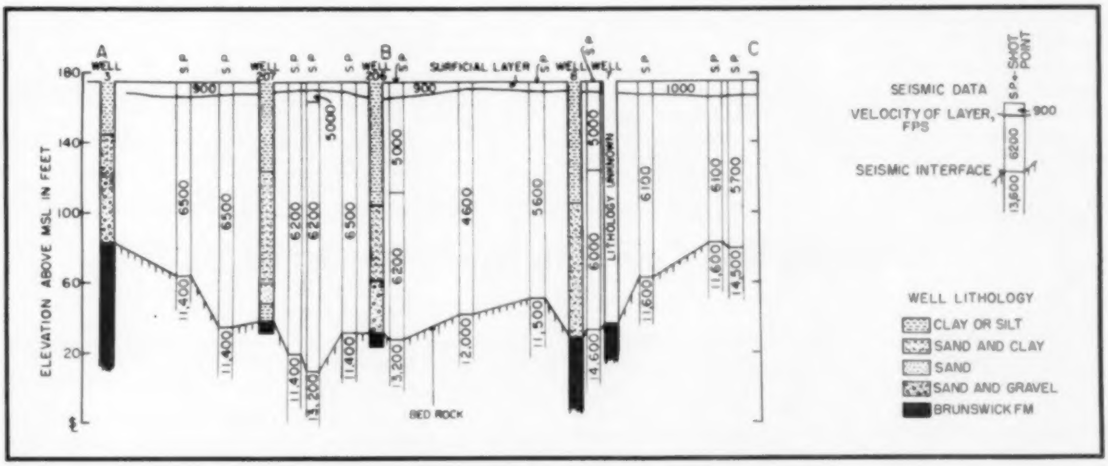


Fig. 4—Cross section A-B-C showing close comparison between well data and seismic-predicted bedrock elevations.

10 ft of surface and is correlated with unconsolidated valley fill below the water table. In five of the spreads two intermediate velocities were recorded. There are not enough data to correlate definitely the higher of these velocities with lithology. However, the log of test well 206 (Fig. 4) located near a shot point indicated a change from predominantly silt to inter-stratified *sand and clay* at a depth very near the velocity interface. Similarly, two other spreads near well 205 recorded velocities of 4500 and 6100 fps in one case and 4700 and 6800 fps in the other. The log of well 205 shows a change in lithology from gray clay to fine-to-medium gravel within 5 ft of the depth of velocity transition.

As can be seen, there is a very favorable velocity contrast between the unconsolidated valley fill and the bedrock in this area. The thickness of material above bedrock ranged from a minimum of 37 ft to a maximum of 159 ft.

Contours on the bedrock have been drawn (Fig. 1) on the basis of the seismic information and from well data. The main buried valley, in which the existing production wells are located, narrows and branches to the north. In the south the valley widens considerably where it is joined by a tributary buried valley entering from the northeast. Cross section A-B-C (Fig. 4) shows the close agreement between seismic and well-log data. Wells 206 and 207 were drilled at two locations indicated by the seismic work to be in the main valley. Wells 3, 7, and 8 existed prior to the seismic work.

A pumping test conducted at test well 206 indicated that a permanent production well at this site would be as good as any of the existing production wells or better. A new production well is scheduled for this location. Well 207 was drilled 800 ft west of well 206 toward an existing production well. However, the aquifer material at this site was too thin to warrant construction of a production well. The northeast branch valley was not explored in detail since it has been observed that maximum yields are obtained from wells located near the Passaic River. A single test well drilled in the area (well 211) did not encounter good aquifer material.

Table I lists actual bedrock depths determined from six test wells, with bedrock depths predicted by nearby seismic work. The predicted seismic depths given in the table for each well site were projected from nearby seismic data. At well 206,

Fig. 1 (left)—Contour map of Triassic bedrock surface in part of the East Orange Water Reserve. Well and seismic control are indicated.

drilled at the exact site of one of the seismic depth determinations, bedrock was encountered within 3 ft of the predicted depth. Maximum error determined was 11 pct; mean error was about 7 pct.

Table I. Comparison of Actual Depths to Bedrock Determined by Test Drilling with Those Predicted by Seismic Survey

Well No.	Actual Bedrock Depth, Ft	Predicted Seismic Depth, Ft	Percentage Error
8	140	137	− 2
202	60	55	− 8
203	51	47	− 8
204	110	122	+11
206	138	141	+ 2
211	121	131	+ 8

Conclusions

The satisfactory results of the seismic survey made in the East Orange Water Reserve show the desirability of conducting a seismic survey in any ground-water exploration program where determination of the thickness of unconsolidated water-bearing deposits is important. In the studies described above, considerable expense and time were saved by the seismic survey. An average of five seismic spreads and from 10 to 16 depth determinations were made each day. By contrast, drilling a 6-in., 100-ft cable-tool test hole took from two to five days and provided only a single depth determination. It is estimated that the cost of the 69 seismic depth determinations was comparable with the expense of drilling a single 100-ft test well. The seismic survey did not eliminate the need for all test wells but did define the most favorable areas in which to drill test wells to obtain critical aquifer information.

Acknowledgments

The writers wish to thank Charles G. Bourgin, engineer and general manager, East Orange Water Dept., and Leggette, Brashears & Graham, Consulting Ground-Water Geologists, for permission to use the information contained in this report.

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[2] M. Ewing, G. P. Woollard, and A. C. Vine: Geophysical Investigation in the Emerged and Submerged Atlantic Coastal Plain, Part III—Barnegat Bay, New Jersey Section. *GSA Bull.*, 1939, vol. 50, pp. 257-296.

Discussion of this paper sent (2 copies) to Aime before April 30, 1958, will be published in MINING ENGINEERING.

Technical Note

Cleaning Fine Coal with Newly Developed Jig

by E. H. Citron

CLEANING fine coal in jigs is not new in Europe, where the feldspar jig is being used almost exclusively for this purpose.

A feldspar jig operates with an artificial bed made up generally of feldspar or other hard rock material. Dimensions and specific gravity of this material must be of suitable values for good bedding qualities. For good efficiency, the size should range between 1x2 in. and 2x2x½ in. thick.

The disadvantage of an artificial bed is that it is not mobile and must be replaced every so often with new material to obtain good cleavage characteristics. A semi-mobile bed has been developed to retain only a part of the feldspar on the sieve screen—the other part is drawn off with the refuse material, from which it is reclaimed and returned to the jig feed. Recirculating this feldspar rock offers definite advantages. However, in time it will lose its original shape because of wear, and new bed material must be added to the jig feed.

The latest feldspar jigs are generally air-pulsated, Baum-type jigs, although more than 70 pct of the feldspar jigs operating in Continental Europe are still of the piston type. These jigs are very popular in Europe, but they do require a great deal of time and attention.

New Theory to Eliminate Artificial Bedding: At the preparation plants of Pittsburg & Midway Coal Mining Co. it has always been the contention that a small piston-type jig could be developed to clean fine ¼x0-in. coal down to 28 to 60 mesh efficiently by obtaining and admitting bedding material from the present plant refuse circuit to the jig feed. This eliminates use of artificial bedding materials. The jig should incorporate these essential adjustable features as follows:

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TN 475F. Manuscript, March 12, 1957.

Adjustable Sieve Screen

A screen that can be raised and lowered to increase or decrease washing compartment depth. The sieve screen also can be set on an incline level, or decline.

Adjustable Plunger Eccentric

Length of stroke can be adjusted to increase or decrease pulsion and suction.

Variable Plunger Speed

Variable speed reducer for decreasing and increasing the number of strokes per minute.

Water Distribution

Adjustable baffle plate separating the plunger and sieve compartment so the water can be distributed uniformly over the entire sieve screen area.

Refuse Ejector

Special unique rubber-type refuse ejector gate operated by air and electrically controlled (as a conventional jig).

Automatic Control

Conventional float that measures the specific gravity of the coal-water refuse medium in the jig is of the submerged type. The float is mounted on a vertical stem held in position by two parallel arms. A movable weight regulates the effective specific gravity of the float. The float stem can be raised and lowered, having a graduated scale to indicate the depth of bed for the setting of the float.

Hutch Drawoff

Hutch material is drawn from the cell compartment to the elevator compartment by gravity.

The first one-cell experimental jig incorporating all these features was built in January 1955 by Ore

Reclamation Co. of Picher, Okla., under the direction of Pittsburg & Midway Coal Mining Co. The experimental jig was installed at the Pittsburg & Midway mine No. 19 preparation plant in Kansas, where it was set up to handle from 20 to 22 tph of material consisting of underflow from the five 14-in. cyclones and the refuse from the Rheolaveur ⅛x0-in. fine coal washer. These two combined materials contained 45 to 55 pct of −28 mesh material, 50 pct of which floated at 1.40 sp gr, with a dry ash content of 10.2 pct. The total raw feed contained a 38.2 pct dry ash.

Results obtained from this experimental jig were above expectation. With respect to size, the jig performed most efficiently on all the sizes above 35 mesh—these sizes respresented 75 to 80 pct of total jig product.

Experimental Jig: The company experimented with several combinations of number of strokes per minute and length of stroke, ranging from 200 to 50 strokes per min with length of strokes ranging from 2 to ¾ in. Best results were obtained with the jig operating at 70 strokes per min, having a stroke length of 1 in. This combination of number of strokes per minute and length of stroke does not agree with theories presented in technical papers on fine coal jigging or with authorities on jigging. For treating ⅛x0-in. coal with a plunger-type jig, the revised formula would be as follows:

T = total travel of plunger
S = strokes per minute
L = length of stroke in inches
$T = (2 \times S) \times L = 140$

For example:

If 70 strokes per min are made, the travel of the plunger per revolution of the eccentric will be $140 + 70 = 2$ in., and the stroke will be $2 + 2 = 1$-in. stroke.

Length of stroke:

The amplitude of pulsation is expressed by the following formula, attributed to Rittinger:

$$V = NA\frac{3.14}{60} \qquad A = V \times N + 60 \times 3.14$$

Table I. Performance Data for New Fine Coal Jig

Test No. 1

Jig Feed: ⅓ mm x 0, consisting of underflow from seven 14-in. cyclones together with ⅛x0-in. refuse from Rheolaveur fine coal washery.
55 pct 1.40-sp gr float in feed
ash of 1.40-sp gr float
capacity, 22.0 tph
raw ash, 38.3 pct
dry ash, 7.5 pct

Jig Product Analysis:

Mesh	Wt, Pct	Dry Ash	Accumulated Ash
+6	2.2	5.1	5.1
6 x 8	3.4	5.3	5.2
8 x 10	2.4	5.7	5.3
10 x 18	19.1	6.2	5.9
18 x 35	44.8	8.1	7.2
35 x 60	24.2	17.0	9.7
60 x 100	3.4	26.0	10.2
−100	0.5	34.6	10.4

Jig Refuse: 10 pct 1.40-sp gr float; dry ash on float, 8.5 pct

Test No. 2

Jig Feed: same as for Test No. 1
56.5 pct 1.40-sp gr float in feed
30.4 pct raw ash
10.21 pct dry ash on 1.40-sp gr float
capacity 24.0 tph

Jig Product Analysis:

Mesh	Wt, Pct	Dry Ash	Accumulated Ash
+6	2.6	8.55	8.55
6 x 8	3.8	9.27	8.90
8 x 10	2.5	9.38	9.20
10 x 18	19.3	10.11	9.8
18 x 35	42.9	11.2	10.6
35 x 60	14.7	16.28	11.6
60 x 100	12.9	22.7	13.0
−100	1.4	30.05	13.3

Jig Refuse: 11.6 pct 1.40-sp gr float; refuse contained 11.23 pct ash on 1. 40 float; refuse raw ash, 45.93 pct

Discussion

Relation of Magnetic Susceptibility to Mineral Composition

by Ernest M. Spokes and David R. Mitchell

(MINING ENGINEERING, page 373, March 1958, vol. 211)

S. C. Sun: This article by Spokes and Mitchell deserves high commendation. For many years mineral dressers have been at a loss to explain the variation in magnetic susceptibility of the same mineral species obtained from different localities. In fact, they do not even have a list of accurately measured data on the specific magnetic susceptibility of weakly magnetic minerals to consult. Experience as a university teacher has firmly convinced me that the average student has more difficulty in acquiring a satisfactory knowledge of magnetism than of any other branch of mineral dressing. This is almost entirely because the subject has been treated far too much from a strict practical standpoint and far too little from a fundamental one. It is no exaggeration to state that almost all the published theoretical discussions are based on the experimental results of manufactured chemicals and alloys, but not of naturally occurring minerals. In an attempt to remedy this state of affairs, the author of this article not only set up a new magnetic balance in the Department of Mineral Preparation of The Pennsylvania State University, but also performed work of fundamental value, thus pointing the way to further research. This is evidenced by the fact that an investigation of the magnetic susceptibilities of manganese minerals was recently completed by S. G. Watt under the direction of David R. Mitchell, now acting dean of the College of Mineral Industries of the Pennsylvania State University. D. J. Cook is studying the magnetic susceptibility of titanium minerals under my supervision. It is hoped that our research program can be continued for several years and that similar investigations will be initiated at other universities and research institutions.

Ernest M. Spokes *(author's reply)*: The discusser has touched the heart of a pedagogical problem in the field of minerals beneficiation—students do not understand magnetism and, as a result, neither do the graduate engineers. Only recently has the average course in engineering physics started teaching the fundamentals of magnetism, as they are understood in modern physics, in addition to the classical treatment of the mathematical aspects of magnetic phenomena. It is to be hoped that this increasing instruction in fundamentals will improve our understanding and inspire research to determine more of the basic magnetic data that are essential if we are to see improved methods of separation by magnetism.

Intergranular Comminution by Heating

by J. H. Brown, A. M. Gaudin, and C. M. Loeb, Jr.

THE object of most size reduction operations in the mineral industry is to liberate the grains of valuable minerals in the ore from those of the gangue. This is usually accomplished by crushing and grinding the entire mass of ore until there is only a small probability that any single particle contains more than one mineral. During this size reduction only limited control exists over size or composition of the particles exposed to the breaking action, and there is no control over the paths followed by cracks generated during the operation. This lack of control usually results in overgrinding and in production of large quantities of very fine material. The first detriment, overgrinding, is costly in itself, but when combined with the second factor it is doubly so. Not only is the fracture of a free particle unnecessary—the fracture of these particles may also make subsequent separation operations difficult, inefficient, and wasteful.

It has been pointed out[1] that if the object of size reduction is to liberate the valuable mineral component of the ore then, ideally, fracture should follow intergranular paths to the exclusion of transgranular ones. This would result in liberation of the valuable minerals with as little size reduction as possible. This ideal comminution operation is referred to as *intergranular comminution,* and it was the object of the investigation to determine the extent to which it could be developed by heat treatments.

There are many indications in the literature that heating rocks prior to crushing may be favorable. Reports by Holman,[2] Yates[3] and Myers[4] are pertinent. These investigators showed that heating certain rocks prior to crushing them did, in fact, improve their crushing characteristics in that fewer fines were produced, although the fact that intergranular comminution was being effected apparently was overlooked.

In addition, Sosman noted that if there is appreciable anisotropism in the thermal coefficients of expansion of even a pure mineral, then considerable permanent separation of the grains of the rock can be expected as a result of heating the rock to a high temperature.[5] By the same token, if there are appreciable differences in the thermal expansion coefficients of the various minerals of a multi-component rock, similar results should be obtained by heating this rock. This has been tested, partially, by Brenner,[6] who obtained patents covering the heat treatment of some pegmatitic rocks in order to facilitate comminution of these materials. It has also been demonstrated that this may occur in taconite.[12]

Also, the possibility of causing decomposition of one mineral in a rock as a means of promoting intergranular fracture has been considered. Seigle[7] and Schiffman et al.[8] have obtained patents on such processes as applied to calcareous iron ores.

These reports all indicate that heat treatments prior to crushing may contribute materially to intergranular comminution, but they also indicate that no organized attempt has been made to determine the controlling factors of the method or to determine its applicability in general. The present article is a report on the initial phase of such an investigation. The authors have reviewed the claims of prior investigators and have attempted, also, to establish the factors that might determine the applicability of heat treatments in the mineral industry.

In this work 2000-g samples of various rocks were heated in a small laboratory furnace and crushing and sizing operations were carried out in standard laboratory equipment. All samples of each rock were as nearly identical as possible in particle size, grain size, and composition and contained only lumps coarse enough to contain many grains each.

Tests on Granite

A number of tests were made on a coarse grained Finnish granite obtained in the form of coarse chips from a local monument yard. This rock exhibited little variation from piece to piece in either composition or grain size. The minerals contained were quartz, orthoclase, small amounts of hornblende, and minute quantities of mica. Grain size ranged from about 1 mm to about 3 mm.

Temperature of the Heat Treatment: In some cases the granite was heated to a particular temperature and crushed, hot, immediately upon withdrawal from the furnace—in others the rock was allowed to cool before crushing, but without quenching to room temperature after heating. In most tests on granite the heating period was about 2 hr with the furnace at the highest temperature for about 1 hr. Cases in which these periods were varied greatly will be presented separately.

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TP 4731B. Manuscript, Oct. 15, 1957.

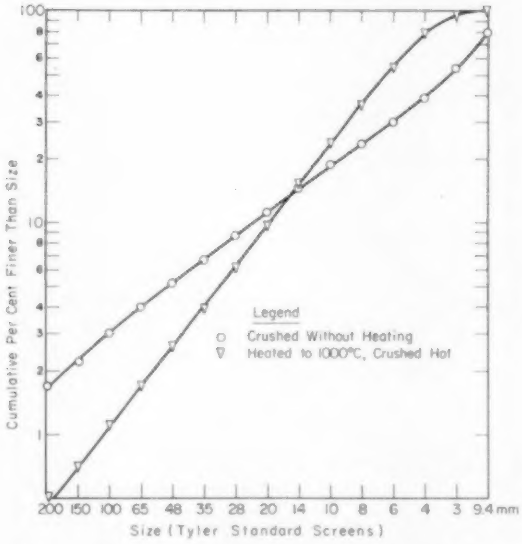

Fig. 1—Size distribution data for granite tests as Schuhmann plots.

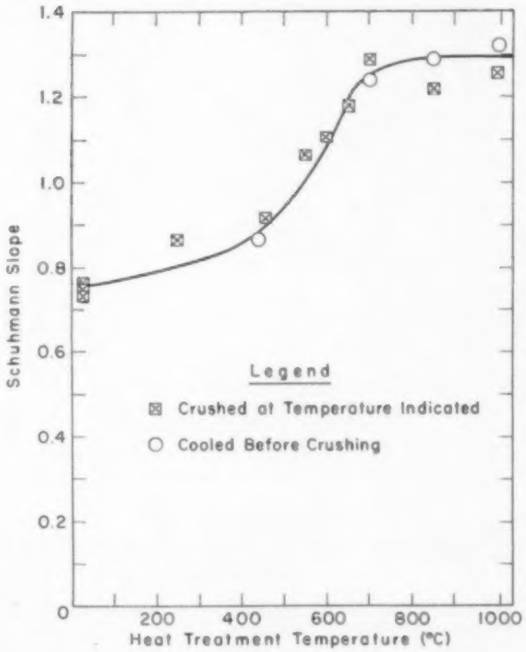

Fig. 2—Slope of Schuhmann curve vs heat treatment temperature for granite tests.

The screen analyses of the various tests were plotted in different ways, but the method illustrated in Fig. 1 was adopted as most suitable for these tests. In this figure the size distribution data for a sample crushed at room temperature and with no heat treatment in a laboratory jaw crusher are plotted as suggested by Schuhmann[9] with similar data for a sample that was heated to 1000°C and crushed at that temperature in the same crusher included for comparison. The difference in slopes of the two distribution curves is striking. The slope of the lines, in their straight portion, is a measure of the dispersion of the crushed product throughout the various size ranges. A high slope corresponds to a product showing little dispersion as in a closely sized product, and a low slope corresponds to a product with a scatter of particles in all sizes. In this case the bulk of the product from tests which had involved heating prior to crushing seemed to concentrate in the size ranges corresponding to the natural grain size of the rock. This was taken as an indication that intergranular fracture was occurring and the slope value was adopted as being indicative of the extent to which such fracture had occurred.

Fig. 2 shows the slope values obtained from plots of size distribution data as a function of the maximum temperature of the heat treatment. The figure shows that a marked change in the size distributions resulted from the heat treatments and that the change due to heating increased with increasing temperature up to a temperature of at least about 700°C. At temperatures above 700°C little further change occurred.

Temperature during Crushing: The data of Fig. 2 are divided into two sets. One series of points represents data obtained when the sample was crushed at elevated temperatures; the other represents that obtained on samples cooled before crushing. It will be noted that while there may be a tendency for samples crushed after cooling to show a different size distribution than those crushed hot (other conditions being the same) the difference appears to be of the same order as the experimental accuracy and is not consistent in its direction.

Tests for Fragility: Two samples of granite, one heated and the other not heated, were crushed in the laboratory jaw crusher set at ⅜ in. and then ground through 20 revolutions of a laboratory ball mill. Size analyses were then made on the products. The data are shown as weight frequency plots on logarithmic scales in Fig. 3. Data for parallel tests that were not ground are included for comparison. The curves show that even the very short grind affected the heated material appreciably, but that the unheated material was almost unchanged except for some production of very fine slimes. The curves show that whereas grinding the heated rock resulted primarily in the breakdown of coarse particles, grinding the raw granite resulted only in production of very fine material without major change in the coarse fractions.

Feldspar Liberation Analysis: The amount of free feldspar in the fraction of each product coarser than 35 mesh is shown in Fig. 4 as a function of the heat treatment temperature. These results were obtained by fractionating each product in a heavy liquid of specific gravity 2.580 ± 0.004. In such a liquid free feldspar particles (of the composition that prevailed in the granite under test) were the only particles that would float; locked particles and free particles of other minerals sank. These data show a large increase in liberation as a result of the heat treatment. Further, the important factor is the heat treatment temperature, not the temperature at which crushing was carried out.

Controlled Atmosphere Tests: One test was run with a nitrogen furnace atmosphere. The gas was passed into the furnace during both the heating and the subsequent cooling steps so that the charge attained a temperature of 1000°C and then cooled without the usual contact with oxygen. The product obtained under these conditions was almost identical to that obtained by heating to the same tem-

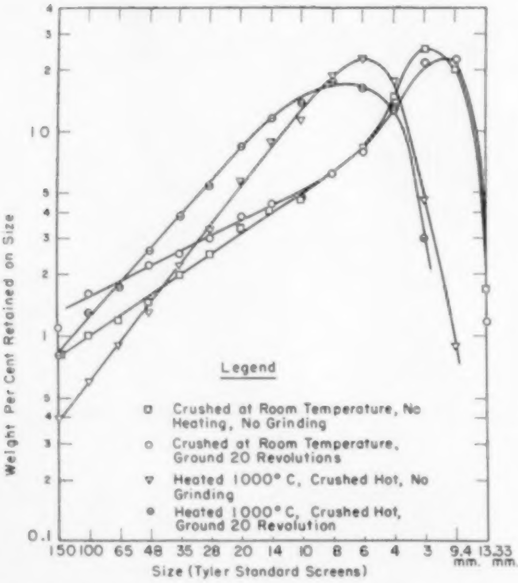

Fig. 3—Frequency plots of size distribution data for tests for fragility of granite after heating.

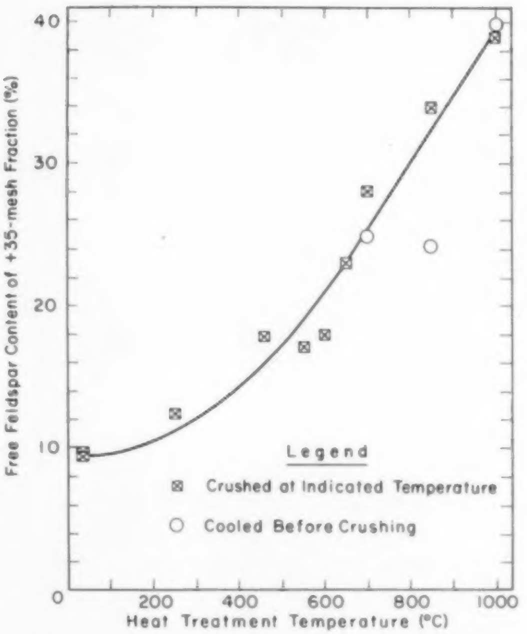

Fig. 4—Free feldspar content of 35-mesh material vs heat treatment temperature tested for granite tests.

perature in air. This similarity indicates that the substitution of nitrogen for air had no effect and that the cause of the disintegration probably was not an oxidation reaction.

Long Heating Tests: In previous tests the charges were merely heated in the furnace for about 1 hr. Two additional tests were made, therefore, to determine whether longer heating at lower temperatures could be equivalent to a higher temperature treatment. Test details are indicated in Table I. Data for tests at shorter heating times are included for comparison.

Table I. Operating Conditions and Results for Long Heating Tests

Treatment Temperature, °C	Time at Temperature, Hr	Size Distribution Slope
450°	6	1.03
500°	6	1.09
460°	1 (approx)	0.92
700°	1	1.25

Temperature control during the heating period was considered satisfactory with an estimated maximum variation of ±25°C.

Slope value in the 6-hr test at 450°C is slightly higher than that obtained for the parallel 1-hr tests, but the difference is of the same order as the experimental accuracy. Also, the slope for the 6-hr test at 500°C did not approach the value for the 1-hr 700°C test.

Grindability Test: The grindability of an ore is evaluated customarily by determining the amount of material finer than a given size which is produced by a reproducible standard grinding operation under steady state conditions. For the granite used in these tests 48 mesh was selected as the splitting size, since it was believed that this was the coarsest size at which the unheated and ground granite would be completely liberated. For this work, therefore, grindability was defined as the steady state amount of −48 mesh material produced during 100 revolutions of the mill.

The test on heated rock was made as follows:

1) Several charges were heated to 1000°C and cooled before crushing.

2) One crushed product (2000 g) was ground in a laboratory ball mill through 100 revolutions of the mill.

3) After the entire ground product was sized, the −48-mesh fraction was set aside, replaced by an equal weight of heated and crushed but not ground product, and thoroughly mixed.

4) The remixed material from step 3 was ground through another 100 revolutions of the mill.

5) Steps 3 and 4 were repeated until the amount of −48 mesh material set aside reached a constant value.

A second test was run in the same manner as this first one except that no heat treatment was used on either the original or make-up material. Only four grinds were made in this case, since the pattern of the first test indicated that a satisfactory measure of the grindability had been obtained by them. The results of the two tests are given in Table II. The data indicate grindabilities of about 340 g per 100 revolutions for the heat-treated granite and about 170 g per 100 revolutions for the raw granite. As indicated in the table there was no tendency for the slope values for the tests on the two different materials to approach the same value as the grinding treatments continued. The coarse fractions of the heated rock continued to break with relatively fewer fines than unheated rock.

It is fair to call attention to the fact that the grindability tests that were made actually penalize the quality of the grinding following heat treatment. In spite of this penalty the grindability after heat treatment showed to great advantage, as already indicated. The penalty consists in this, that the

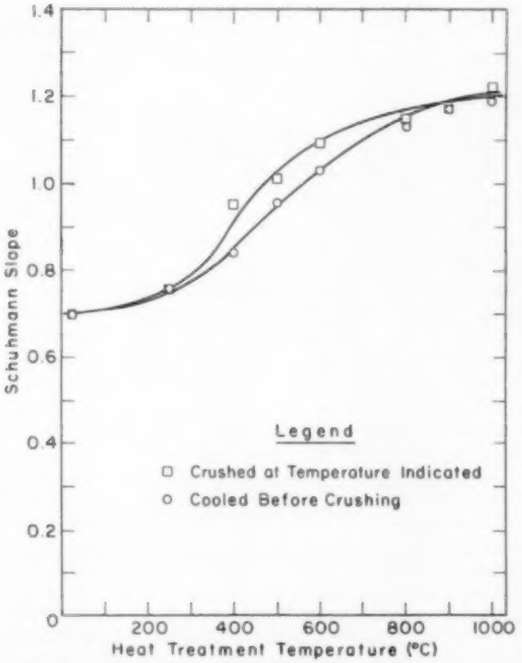

Fig. 5—Slope of Schuhmann curve vs heat treatment temperature for syenite tests.

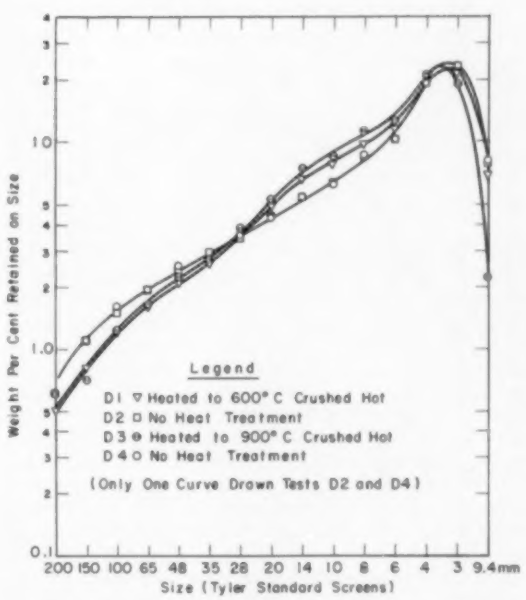

Fig. 6—Frequency plots for size distribution data of tests on franklinite.

selection of 48 mesh as the size to which the grindability was assessed, while suitable from the standpoint of the unheated rock, is unsuitable from the standpoint of the heated rock, since most of the grains have been freed by crushing after heating at a size substantially coarser than 48 mesh. Had the assessment size of the grindability been, say, 10 mesh, the writers are convinced that the ratio of the grindabilities of heated and unheated rock would have been much greater than 2:1. Even such a comparison would not have been appropriate. What really is called for is a comparison of the grindabilities of heated rock at, say, 10 mesh with unheated rock at 48 mesh. There the ratio of the grindabilities might easily have exceeded 10:1.

Table II. Experimental Data for Grindability Tests on Granite

Test Material	Grind No.	Product Grams of −48 Mesh	Slope Value of Product
Heated Granite	1	296	0.86
	2	303	1.03
	3	321	1.12
	4	331	1.18
	5	340	1.19
	6	343	1.20
	7	351	1.20
Raw Granite	1	200	0.73
	2	197	0.84
	3	184	0.87
	4	171	0.86

Tests on Syenite

It was considered that the presence of quartz in granite might have been directly responsible for its crushing behavior after heating, and that this in turn was due to the alpha-beta phase transformation of quartz which occurs at about 575°C. A syenite was chosen for testing as a material close to granite in composition and texture but containing very little quartz.

Approximately 200 lb of syenite were obtained from the Mt. Ascutney area of Vermont. This material exhibited a constant grain size of about 1.5 mm and contained orthoclase, plagioclase, hornblende, nepheline, and small amounts of magnetite and quartz.

The tests made on syenite involved heating samples to various temperatures and subsequently crushing them with the jaw crusher set to a minimum opening of 9 mm. In all heat treatments the charges were in the furnace approximately 2 hr, and as with the granite tests, half this time was spent in raising the furnace to temperature and about half in allowing the charges to remain at that temperature. In some cases the heated rock was crushed hot and in others it was allowed to cool before crushing.

Data from all size analyses were plotted in the same way as with granite, and the slope values were determined from these plots. In Fig. 5 the slope value for each test is plotted as the ordinate against the heat treatment temperature as abscissa. The curves so obtained show that the heat treatment brought about changes in size distribution comparable to the changes observed with granite. In addition, it appears that the slope values for products crushed after cooling lie slightly lower than those obtained for products crushed at elevated temperatures.

The significant conclusion to be drawn from Fig. 5 in relation to the purpose of this work is that the presence of quartz is not needed for heat treatment to produce a marked favorable effect on crushing.

Tests on an Isotropic Mineral

Among the ores and rocks on hand there appeared to be only one that was suitable for heating tests and consisted primarily of one isotropic mineral. This was a rich ore from Franklin, N. J., the major

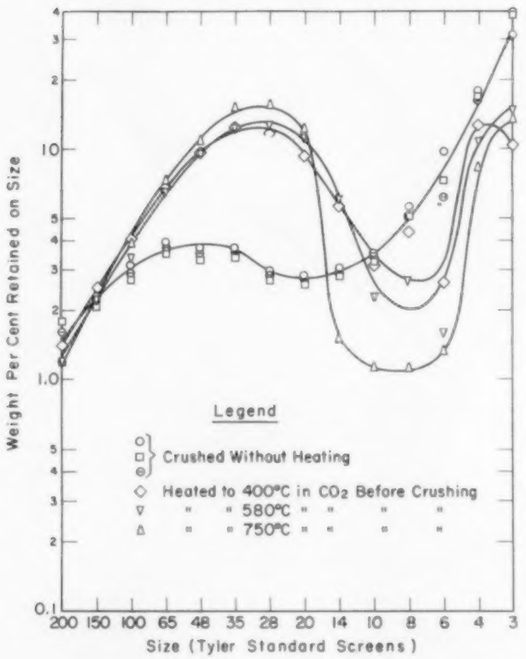

Fig. 7—Frequency plots for size distribution data of marble tests.

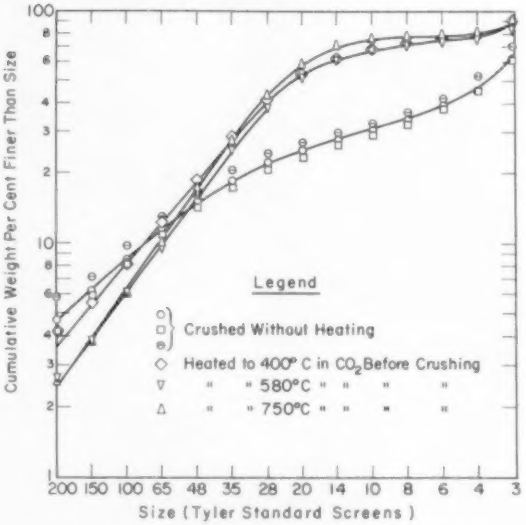

Fig. 8—Size distribution data for marble tests as Schuhmann plots.

constituent of which was the isotropic mineral franklinite. Only enough of this ore was available to provide four 1500-g charges of −35-mm + 12-mm material. These samples contained about 5 pct calcite as small veins in the bulk material. Grain size ranged between 0.3 and 0.5 mm.

Two of the charges were crushed at room temperature without any heat treatment, one was crushed at 600°C, and one was crushed at 900°C. The size distribution data are shown as frequency plots on logarithmic scales in Fig. 6. This method of plotting the data was used in this case, since it makes small differences in the distributions more obvious than does a cumulative plot. From these curves it appears that the heat treatment had only a small effect on the size distribution of the material. Since some calcite was included in the Franklin ore there can be no assurance that this was not responsible for the small differences.

Tests on Taconite

Since it was desirable to supplement this work on special systems with tests on systems of practical interest, a sample of a magnetic taconite was obtained from Pickands, Mather & Co. Examination of a polished section and hand specimens of the taconite indicated that much of the iron mineral was present in veins up to about 10 mm in thickness, but that a considerable amount was present as very fine grains ranging down to 0.1 mm and finer, imbedded in the siliceous gangue. This fine grain size indicated that if a difference in the liberation were to be detected the screen analyses would need to be done with the utmost care.

Of the four tests made on this taconite, two samples were heated and crushed at 1000°C and two were crushed without any heat treatment. The jaw crusher was set to a minimum aperture of 2.5 mm.

The size distribution data of the products, when plotted as frequency plots, showed only slight differences in the amounts retained on sizes between 48 mesh and 200 mesh, with slightly more material reporting in these sizes in the heat-treated materials. The differences, however, did not warrant further testing of the rock at this stage of the investigation.

A brief microscopic examination of the fine sizes in the products showed that the liberation was essentially complete at about 150 mesh. It was also apparent that much of the gangue in this ore was optically anisotropic.

Tests on Marble

Marble was selected for testing as a mineral exhibiting strong thermomechanical anistropism. A sample of Danby marble, obtained from the Vermarco Lime Co., consisted of pure white stone exhibiting a regular grain size of about 0.5 mm. It contained 95 pct or more of carbonate mineral.

Table III. Equilibrium Conditions for $CaCO_3$ Decomposition

Temperature, °C	Equilibrium CO_2 Pressure, Atm
550°	0.0057
700°	0.029
800°	0.22
900°	1.0
937°	1.77

Since marble decomposes at high temperatures to give lime and carbon dioxide, heat treatments were restricted in their scope. Published data[18] on the equilibrium decomposition conditions are reproduced in Table III.

From these data it appeared that if marble were heated in a CO_2 amosphere, temperatures to about 900°C should be possible without risk of decomposition. Tests were made, therefore, with the furnace continually flushed with CO_2 during heating and cooling. The charges were heated to temperatures of 400°C, 580°C, and 750°C, and the heated and cooled

products were crushed with the jaws set to a minimum aperture of 6 mm.

The size distribution data for these crushed products are given in Fig. 7 as frequency plots and in Fig. 8 as cumulative plots. These two figures demonstrate the values of the two methods of plotting very well. It will be noted that the heated marble assumed a distribution far different from that which would be expected in a normal homogeneous material. This departure from normality is strikingly shown in the frequency plot but is not so apparent in the cumulative plot. The curves in Figs. 7 and 8 show that the heat treatment had two very marked effects on the size distribution. First, there is a remarkable similarity between the several heat treated products. Since heating to 400°C was almost as effective as heating to 750°C it seems possible that even raising the temperature by 100°C might give a noticeable effect, but this matter was not pursued. Second, it appears that much of the products from tests on heated marble concentrated in size ranges corresponding to the grain size of the rock (20 to 35 mesh). This indicates that much more intergranular fracturing occurred during crushing and screening of the heated rock than with unheated material. It was also noted that it was possible to crush lumps of heated marble to a sugary product merely by pressing between the fingers.

As a check on the extent to which chemical change might have taken place during heating, the −200-mesh fractions of the various products were pulped with distilled water and the pH of the water was then measured. If dissociation had occurred during heating then the CaO so formed should have given a higher pH to a pulp containing heated marble than to one containing unheated marble. Since none of the pulps showed a pH above 9, it is unlikely that decomposition took place during any of the heat treatments. Furthermore, tests at 700°C without atmosphere control yielded pulp pH's of 11.5 whether crushed hot or cold but still yielded almost the same screen analyses as already reported for the test at 750°C. Clearly, the intergranular fracturing of this marble sample was not primarily due to intergranular corrosion but to some other factor.

Discussion

The results of this investigation may be examined with respect to the various mechanisms by which the heating may have affected the rock. The possibility that heating rocks prior to crushing them merely amounts to introducing the energy required for fracturing as heat rather than as mechanical energy can be eliminated on the basis of data obtained. For this mechanism to be responsible the temperature of the rock during crushing should be very significant and the effects might be expected to be independent of the material being tested. Furthermore, if heating merely supplies comminution energy then only parallel translation of the size distribution curves toward the finer sizes without changes in their shapes should result from heating. Since the data do not support any of these requirements, it is logical to assume that the fracture process under investigation here is not the same as that which results when unheated rocks are crushed.

When a metal piece is broken at elevated temperatures or when the fracture is induced by a slow rate of loading, fracture will usually occur intergranularly. Conversely, with a high rate of loading or relatively lower temperatures the fracture will be transgranular and heterogeneous.[10,11] It might be supposed that a similar relation could exist in rocks, but no conclusive evidence was obtained to support this hypothesis. The temperature during crushing was not particularly significant, although it was shown that heating the rock had a marked effect in some cases. The syenite tests did indicate a slight advantage in crushing hot over crushing cold, but this was a small improvement over the improvement due to the prior temperature change.

It was noted in the tests on granite that the effect of heating became large in the vicinity of the quartz alpha-beta transformation (about 575°C). Since this transformation involves a large volume change in the quartz, it appeared possible that that mineral was literally forcing the rock apart by its sudden expansion. With most of the expansive strain focused on the grain boundaries the intergranular nature of the fracture seemed logical. The test on syenite almost duplicated those on granite, however, indicating that while the quartz transformation might have been effective to some extent it certainly was not the only factor. The tests on marble which showed that rock to be markedly affected by heating further reduced the possibility that the quartz transformation plays a dominant part.

The tests on marble were also related to another suggestion. It had been proposed in at least one patent that decomposition of calcite in an ore by heating would promote intergranular fracture of that mineral. If heating of marble were carried out only so far as to weaken intergranular bonds by causing decomposition between the grains, a logical place to expect decomposition to begin, then intergranular corrosion would be achieved. Tests in which the marble was heated with the furnace atmosphere containing more than enough CO_2 to prevent such decomposition did not prevent its breakdown, however, indicating that this intergranular corrosion was not a necessary feature of the heat effect.

It could be argued that a chemical change, for example, oxidation of ferrous iron to ferric iron, was in some way responsible for the different size distributions of the crushed products, but when granite was heated in a nitrogen atmosphere, no change in the heated crushed product could be detected. In addition, it was believed that no chemical change took place in the syenite on heating and it was shown by pH determinations that neither wholesale nor intergranular decomposition of marble had occurred in tests on that mineral. Thus, it seems unlikely that a chemical change in the rock was responsible for the effectiveness of heat treatment.

At least two other possible mechanisms could not be so completely resolved on the basis of the data obtained. The possibility that anisotropism of some of the constituent minerals of the rock was necessary was not proved by the tests on Franklin ore. A small effect was noted in this material when crushing it after heating, but that could have been due to the small calcite content of the rock. Furthermore, the taconite contained an anisotropic siliceous gangue and yet only a very small heat effect could be shown.

If unequal expansion of the individual grains in the grain aggregate were effectively tearing the aggregate apart along grain boundaries, both the intergranular nature of the fracture and the dependence of the fracture on the maximum temperature attained would be expected. Unfortu-

nately, the tests do not permit evaluation of this suggestion, for essentially identical results might have been obtained if the effects noted were due to spalling because of different heat transfer rates through the various grains of the rock. It does appear probable that one or both of these methods contribute to the process, but much work must be done to define their respective roles. In addition, the effect of such features as the grain size and porosity of the rock might bear investigation.

Conclusions

It has been shown that heating a rock prior to crushing it may result in marked improvements in its friability and in the size distribution of the crushed product. The effects noted were of the nature to be expected if intergranular fracture were occurring as a result of heating. The mechanism by which this is accomplished can not be stated with any certainty, but it appears to be more dependent on physical changes in the rock at elevated temperatures than on chemical changes.

The potential value of these treatments appears sufficient to warrant further investigations. In particular, it is the authors' hope that not only will the mechanism by which heating is effective be determined but also that a contribution can be made to the understanding of intercrystalline bonding in brittle materials.

Acknowledgments

The work presented in this article was benefited by association with a project on comminution in course at Massachusetts Institute of Technology. That project is sponsored by the Engineering Foundation, American Iron and Steel Institute, Allis-Chalmers Mfg. Co., the Atomic Energy Commission, Norton Co., International Nickel Co., and Carborundum Co. The assistance and encouragement of these organizations is gratefully acknowledged.

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Abstract

Interpretation of the Literature on the Mechanism of the Hall Process

by John J. Stokes, Jr.

(TP 4589D, Transactions of the Metallurgical Society of AIME, page 75, February 1958, vol. 212, no. 1)

Literature on the electrolysis of aluminum from cryolite melts and on the structure of these melts is surveyed critically. Data on density, freezing point, and other properties are reviewed. Theories of the electrolysis are examined in the light of these data. Two theories are presented which account equally well for the observations.

* * * *

In the early days, the process was thought of as a simple one. The actual mechanisms must be tremendously more complex. This paper is devoted to a rather narrow part of the whole subject, namely, what ions are present in the bath, and by what mechanism the current is carried.

* * * *

Using cryoscopic data, Rolin has determined the following composition for molten cryolite.

$$Na_3AlF_6 \rightarrow 3Na^+ + 3F^- + AlF_3. \quad [1]$$

He postulates the following equation for the ionization of alumina at low concentrations

$$Al_2O_3 \rightleftarrows Al^{+++} + AlO_2^- + O^=. \quad [2]$$

At concentrations above 0.025M he supposes the ionization of alumina to proceed as follows

$$Al_2O_3 \rightleftarrows AlO^+ + AlO_2^-. \quad [3]$$

Rolin's mechanism of electrolysis for the more concentrated solution in Eq. 3 is as follows. The AlO^+ ion migrates to the cathode, where it is discharged, giving metallic aluminum and $O^=$ ions which migrate to the anode. The AlO_2^- ion migrates to the anode, where it is discharged, giving Al^{+++} ions and oxygen which reacts with carbon to form CO_2.

In the case of the extremely dilute solution in Eq. 2 the Al^{+++} ion migrates to the cathode and the AlO_2^- and $O^=$ migrate to the anode as before.

* * * *

Another possible mechanism for the electrolysis follows.

Ionization of Cryolite

$$Na_3AlF_6 \rightleftarrows 2Na^+ + 2F^- + NaAlF_4$$

$$\downarrow\uparrow$$

$$Na^+ + AlF_4^- \rightleftarrows AlF_3 + Na^+ + F^-. \quad [4]$$

Addition of Alumina

$$2Na_3AlF_6 + 2Al_2O_3 \rightleftarrows 6Na^+ + 3Al_2O_2F_4^= \quad [5a]$$

$$Al_2O_2F_4^= \rightleftarrows 2AlOF_2^-. \quad [5b]$$

Cathode Process

$$AlF_4^- + 3e \rightarrow 4F^- + Al. \quad [6]$$

Anode Process

$$AlOF_2^- + 2F^- \rightleftarrows AlF_4^- + 1/2O_2 + 2e$$

or

$$Al_2O_2F_4^= + 4F^- \rightleftarrows 2AlF_4^- + O_2 + 4e. \quad [7]$$

* * * *

Although both Rolin's and the present mechanism of the Hall process may be qualitatively satisfactory, present data are not adequate for quantitative tests of the theories.

SME BULLETIN BOARD

Reports of Your Technical Society

Newsletters . . .

Rock in the Box .... page 499

Industrial Minerals .... page 500

Don't forget these meetings . . .

Pacific Northwest Regional

AIME-ASME Joint Solid Fuels

Uranium Section Annual Symposium

see page 498

Progress . . .

on new United Engineering Center, see page 449

Coming Next Month

Full Report on SME's role at the 1958 Annual Meeting

For Fun . . .

Be sure to see page 507 and a report on Utah Section's February smash hit.

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CHANGE OF ADDRESS: To be sure that you receive your publications and correspondence promptly please fill in the form below and send it to **Society of Mining Engineers of AIME**, 29 W. 39th St., New York 18, N. Y.:

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Any recent activity that would be of interest to members:

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Field Trips, Sessions Planned for May At Moab Uranium Meeting

Technical sessions and field trips through the mill of Uranium Reduction Co. and the mines of the Big Indian District in San Juan County, Utah, will highlight the third annual Uranium Symposium sponsored by the Uranium Section. The Symposium, under the direction of Philip Lindstrom, will be held in Moab on May 9 and 10 with field trips on the 11th.

On the social side, the feature of the meeting will be a "repeat performance." Once again arrangements have been made for the Chuck Wagon Dinner to be held at Charles Steen's fabulous home and estate overlooking Moab.

The technical program has been divided into three sections: geology, mining, and metallurgy, with two symposiums climaxing the proceedings. One symposium will be a joint one for mining and geology and the other will be for metallurgy. The latest program (at MINING ENGINEERING press time in mid-March) is given below.

Responsibility for the geology program goes to Max Pierson and Wm. McDougald and for mining to John Mullen and Wm. Franklin. Plans for the metallurgy program are under the direction of Theodore Izzo and Gerald Stocks.

Geologic Program

Friday, May 9, pm

Geochemical Studies in the Big Indian District, San Juan County, Utah: *Vance Kennedy*, USGS

Collapse Along the East Flank of the Spanish Valley Anticline, Grand and San Juan Counties, Utah: *Willard Puffett*, USGS

Relationship Between Paradox Basin Oil to Structural Control: *Dane Picard*, American Stratigraphy Co.

Saturday, May 10, am

Ore Deposits of the Big Indian District, San Juan County, Utah: *Wm. B. Loring*, Hidden Splendor Mining Co.

Relationship of Paradox Basin Oil to Stratigraphy: *Kenneth E. Carter*, Geologic Consultant, Durango.

Mining Program

Friday, May 9, pm

Mining Practice in the Ambrosia Lake Area: *Ray Schultze*, Rio Di Oro Mining Co.

Ore Control and Mining at the Lucky Mc Mine, Gas Hills Area, Wyoming: *K. G. Wallace*

Saturday, May 10, am

Ore Occurrences of the Front Range Including the Schwartzwalder Mines: *Chas. O. Parker*, Denver.

Continentals' Review of Three Phases of Uranium Mining: *Maurice Brady* and *John Roscoe*, Continental Material Inc.

Geology and Mining Symposium

Saturday, May 10, pm

Ed Snyder, Moderator

Economics of Crushing and Blending at the Cord Mine: *Sam Craig*, Jen. Inc.

Pillar Removal at Mi Vida Mine: *Virgil Bilyeau*, Utex Exploration Co.

Blending of Uranium Ore at Hidden Splendor Mine: *Ned Wood*, Hidden Splendor Mining Co.

Underground Dry Drilling Experiments at the Radon Mine: *Lloyd Fenske*, Hecla Mining Co.

Long Hole Drilling Experiments at the La Sal Mine: *Gordon Miner*, Homestake Mining Co.

Gamma Ray and Electric Logging and Drift Surveying: *A. Nichols*, Century Geophysical Co.

Underground Haulage at Standard Uranium Mine, San Juan County, Utah: *Russell Woods*, Standard Uranium Co.

Metallurgy Program

Friday, May 9, pm

From Ore to Green Salts: *Harry Gardner*, National Lead Co.

Paper on Uranium Metallurgy: *Speaker* from Union Carbide Corp.

Saturday, May 10, am

Instrumentation in Uranium Mills: *Carl Marquardt*

Uranium Recovery by Solvent Extraction: *J. D. Lewis*, Vitro Uranium Corp.

Metallurgy Symposium

Saturday, May 10, pm

Lewis Painter, Moderator

Advantages of Better Cycloning: *Robert Curfman*, Uranium Reduction Co.

Crushing of Wet Ore: *Speaker* to be announced

Solvent Extraction Special Problems: *Speaker* to be announced

Use of Chloride Elution and Resin-in-Pulp: *Speaker* to be announced

Participation Breakdowns in Uranium Milling: *Speaker* to be announced

Special Problems in Uranium Sampling: *Speaker* to be announced

Plans Being Completed For Pacific Northwest Conference in Spokane

Two days of technical sessions and a day of field trips is the agenda for the 1958 Pacific Northwest Regional Conference to be held in Spokane, April 17 to 19.

Sponsoring Sections of this annual event are Oregon, Columbia, and North Pacific, and A. E. Weissenborn, a SME Director, is chairman.

D. Ingvoldstad and his program committee have arranged a varied program, opening with a session on education for the mineral industry, that will appeal to members of the divisions of both the Society of Mining Engineers and The Metallurgical Society. On Thursday afternoon there will be simultaneous sessions on geology, extractive metallurgy, and mining. The technical sessions on Friday will cover physical metallurgy, geology, minerals beneficiation, and industrial minerals.

Headquarters for the three-day meeting will be the Davenport Hotel in Spokane with the Thursday morning opening gathering at the Spokane Chamber of Commerce Building. A full program is being planned for the ladies attending with their husbands.

On Saturday two field trips have been planned: to the new plant of Gladding McBean and Co. at Mica, Wash., and to the uranium mill of Dawn Mining Co. at Ford, Wash. The second trip will include a visit to the company's Midnight mine.

Certainly not neglected in the conference plans is the lighter side of the meeting: a gala roster of social events is included on the program. Festivities get underway on Thursday with an opening luncheon at the Spokane Club. Friday's luncheon will take place at the Davenport as will an informal buffet supper that same evening.

In addition to Messrs. Weissenborn and Ingvoldstad, other members of the three Sections are busily putting the final polish on plans for the April event. R. N. Appling, Jr., is publicity chairman.

1958 Joint Solid Fuels Conference To Be Held At Old Point Comfort

Old Point Comfort, Va., will be the site of the 1958 AIME-ASME Joint Solid Fuels Conference on October 9 and 10. Headquarters hotel will be the Chamberlin.

General conference chairman is E. E. Williams, vice president, Duke Power Co., ASME. AIME co-chairman is George G. Ritchie, Chesapeake and Ohio Railway Co. AIME program chairman is James R. Garvey, Bituminous Coal Research Inc.

Rocky Mount Geologists Plan Symposium, Trip

Rocky Mountain Assn. of Geologists has announced plans for a symposium on the Pennsylvanian rocks of Colorado and adjacent areas to be presented this fall in

(Continued on page 499)

ROCK IN THE BOX

Mining & Exploration Division

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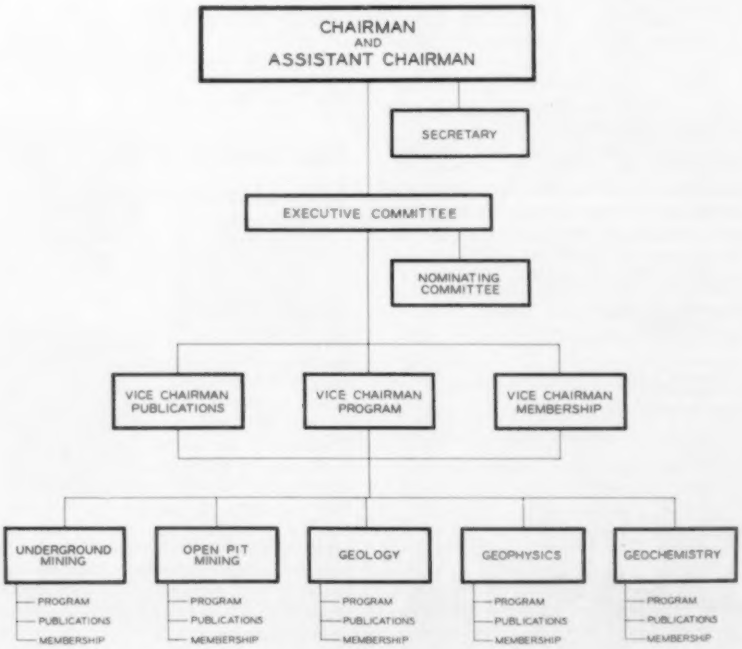
MINING AND EXPLORATION DIVISION



Name John W. Chandler Editor for Rock in the Box

JOHN W. CHANDLER

The new Editor for *Rock in the Box* is on the staff of the Mining Division of American Metal Climax Inc. in New York City. "Jack" Chandler was born in San Francisco and brought up in western mining camps. Graduated from the Michigan College of Mines and Technology in 1925 with B.S. and E.M. degrees he began working at the United Verde Copper Company, Jerome, Ariz. in various jobs up to shift boss. Later he mined gold for 12 years with the Lava Cap Gold Mining Corp., Nevada City, Calif., as general superintendent. After being vice president and manager of the Keystone Copper Corp., Copperopolis, Calif., he spent 11 years with the Eagle-Picher Co. as Director of Mining.

His address is c/o American Metal Climax Inc., 61 Broadway, New York, N. Y.

Rocky Mount Geologists

(Continued from page 498)

guidebook form. A field trip to cover the Pennsylvanian exposures in northwest Colorado will be held in conjunction with the Symposium.

Among the areas to be covered are Pennsylvanian stratigraphy of the Paradox Basin, northwest Colorado, and eastern Colorado. The conference will tour the Maroon Basin of northwest Colorado in order to examine the surface exposures of typical basin-center sediments and typical basin rim or shelf sediments. Particular attention will be given to interesting carbonate and hydrocarbon developments.

A maximum of 200 people can be handled on the caravan-type field excursion. Tentative dates of September 18 to 20 have been set.

INDUSTRIAL MINERALS DIVISION

NEWSLETTER

Dear Members of the Industrial Minerals Division:

Your Newsletter this month is mostly about people and jobs, for it seems to your Secretary that the membership of IndMD should have every opportunity and obligation to belabor and advise its elected representatives. Only by such solicited and unsolicited badgering can we know what you think is good for the Division, the Society, and the Institute. The 1958 officers were listed in MINING ENGINEERING for October 1957, page 1161, and additional appointments are listed below. All the members named are in the Directory so there can now be no excuse for "inactive grass roots."

Because some Division members have indicated as great confusion with the organizational setup as the Secretary freely admitted to last year, here is a flowsheet for our Division, courtesy of Jack Fox, Secretary of the Society of Mining Engineers. *(A flowsheet showing the relationship of the Division to SME appears on the bottom of this page. Ed.)*

1958 Appointments of the Divisional Chairman, R. M. Grogan:

Program Chairman, R. H. Feierabend

Commodity Chairmen

Ceramic Raw Materials—R. F. Brooks

Special Sands and Abrasives—George E. Pettinos, Jr.

Industrial Waters—Leon W. Dupuy

Dimension Stone and Slate—R. A. Fletcher

Chemical Raw Materials—John S. Holland

Cement, Lime and Gypsum—Orville E. Jack

Rare and Radioactive Minerals—Richard H. Jahns

Fillers, Fibers, and Pigments—Lewis R. Moretti

Mineral Aggregates—E. L. Howard

Industrial Minerals Division Members—Society of Mining Engineers Committees:

SME Nominating Committee
Regular members:
A. B. Cummins
Ian Campbell

SME Admissions Committee
L. P. Warriner
Mrs. Pauline Moyd

SME Membership Committee
S. S. Cole
Alternate members:
H. M. Bannerman
R. C. Stephenson

SME Program Committee
R. H. Feierabend

SME Committee on Mineral Economics
J. M. Warde

SME Transactions Editorial Committee
Donald R. Irving

SME General Editorial Committee
John G. Broughton

There will be no regional meeting during 1958 sponsored by Industrial Minerals alone, but the Division will play an important part in making a success of each of the forthcoming regional meetings. We hope that Division officers and members will make every effort to make a contribution whenever possible.

April 17-19—AIME Pacific Northwest Regional Conference, Spokane

September 10-12—AIME Rocky Mountain Minerals Conference, Salt Lake City

October 23-25—Mid-America Minerals Conference, St. Louis

Details on these meetings will appear in MINING ENGINEERING as the time for each approaches.

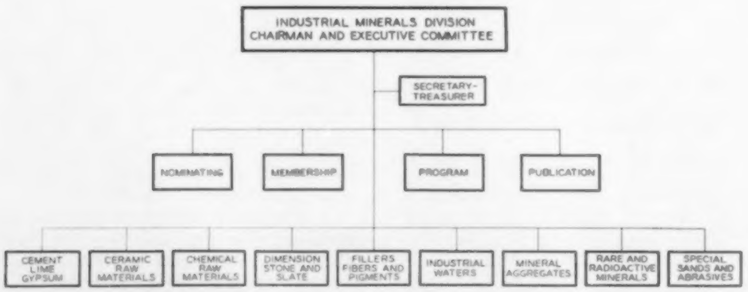
A high point of the IndMD Luncheon on February 18 was the announcement by Mrs. Hal Williams Hardinge of the establishment of a new Industrial Minerals Award to be given for distinguished service in this field of AIME. *(Look for Annual Meeting story in May ME, Ed.)* Joe Gillson and Ray Ladoo were instrumental in the arrangements and planning for this award. A committee of eight has been appointed, to which is added the Divisional Chairman as an ex-officio member, who will accept nominations for the award and decide who may receive it. The first appointees to the committee are:

B. F. Bowie	Richard H. Jahns
Sanford S. Cole	Ralph S. Mason
A. B. Cummins	R. C. Stephenson
M. F. Goudge	Vernon E. Scheid

We hope that every member of the Division will make a very real effort to submit the names of worthy candidates for the consideration of this committee. Many members of the Institute have given long and distinguished service in the cause of Industrial Minerals, and we look forward to the opportunity of honoring them.

John C. Broughton
Secretary-Treasurer
Industrial Minerals
Division, AIME

SOCIETY OF MINING ENGINEERS OF AIME
INDUSTRIAL MINERALS DIVISION



1960 Congress Planned On Mineral Processing

An International Congress on Mineral Processing has been planned by The Institution of Mining and Metallurgy to take place in London, England, Apr. 6 to 9, 1960.

It has been proposed that the program cover fundamental and applied research and development in the fields of mineral dressing, chemical processing, roasting, cyanidation, leaching, and solvent extraction, but not smelting.

Inquiries in regard to the Congress should be sent to: The Secretary, The Institution of Mining and Metallurgy, 44 Portland Place, London W.1, England.

Conference Aimed At Stimulating Research

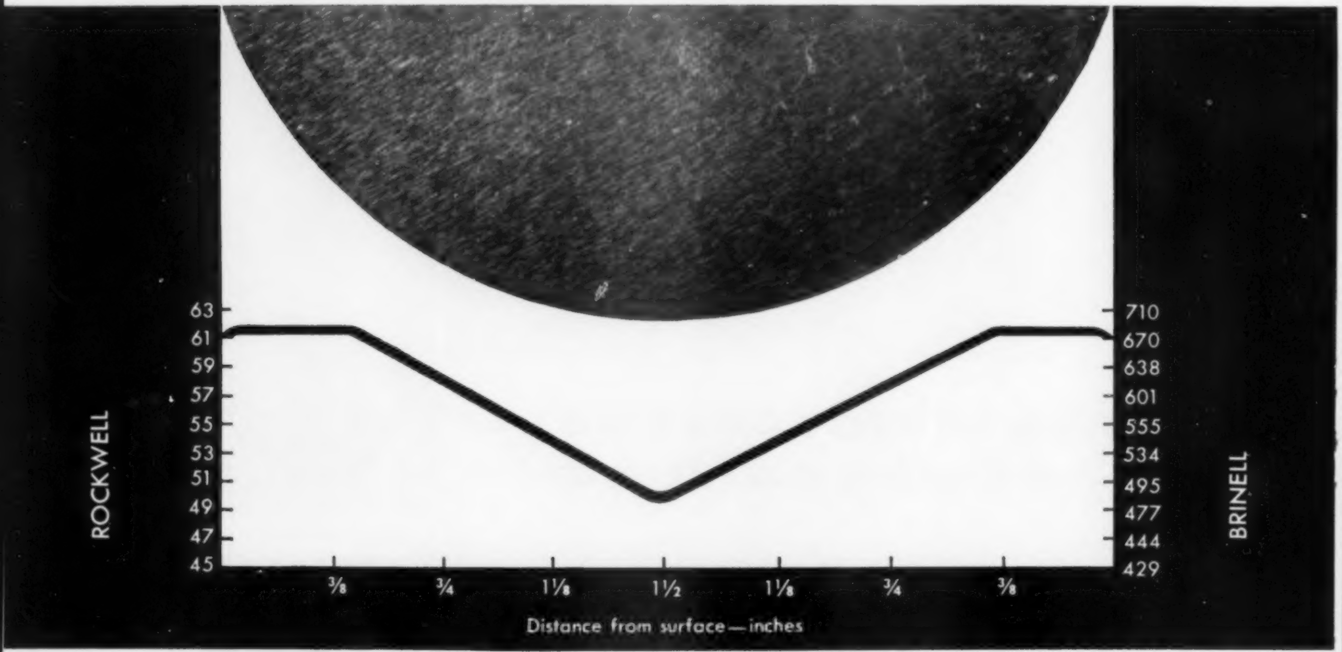
Stimulation of engineering research in smaller colleges and universities was the goal of a conference on February 17 and 18 at the University of Colorado, Boulder. Over 50 engineers from 31 universities in the west were invited to attend the meeting which was sponsored by the National Science Foundation. All sessions were held in the University Memorial Center.

A total of eight sessions were held during the two-day meeting. Among the subjects discussed were: how should research be related to the education program, what kinds of research can best be handled by colleges of engineering, sponsored research—advantages, pitfalls, appropriate sponsorship.

controlled metal hardness means heat-treated cast alloy steel

NACO GRINDING BALLS

cut per-ton grinding costs



Note how new casting process and full heat treatment show controlled hardness between surface and inner core.

Spectrographic analytical control of elements in steel making processes and controlled heat treatment assure the desired metallurgical grain structure which produce the type of hardness required for maximum wearing qualities.

Performance reports on Naco solid cast alloy steel grinding balls from mills now using them have been universally favorable—both in lasting qualities and impact absorption.

Structurally, they possess a grain closely approaching tool steel—tough, hard and rugged for long lasting qualities. Laboratory tests show a remarkable uniformity in solidity, both under X-ray and specific gravity tests, with controlled hardness holding to a desired depth.

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Philadelphia Site Of AIChE Golden Jubilee

AIChE celebrated the start of its golden jubilee year with a Founding Day ceremony on January 29, at the Engineers' Club in Philadelphia, city in which the Institute was born 50 years ago.

Among the highlights of the program was the presentation to the Engineers' Club of a plaque commorating the founding of AIChE in 1908, and the receipt of greetings from Mayor Richardson Dilworth on behalf of the city of Philadelphia. This is to be the scene of a special jubilee convention starting on June 22.

Frederic Dannerth, one of the two surviving members of the original chemical group, recalled the founding events for the approximately 200 scientists and guests who attended the ceremonies. Other speakers were James Harper, Sun Oil Co., who presented a review of chemical engineering progress in the last 50 years by Raymond P. Generaux; and George E. Holbrook, newly elected president of AIChE, who discussed the future of the Institute and the large part it will contribute to national scientific development.

AIME NEWS

AIME

BOARD OF DIRECTORS

Recent actions taken by the Institute Board of Directors.

► AIME staff change announced on February 18: John Lynch succeeds P. J. Apol as Assistant Treasurer upon the latter's resignation effective March 1. Mr. Lynch joined the AIME staff in 1954 as Chief Accountant.

► Announcement was made of the establishment of the Hal Williams Hardinge Award by Mrs. Bertha Hardinge (Mrs. H. W.). This will be given in recognition of outstanding achievement in industrial minerals. For further information, see the Industrial Minerals Division *Newsletter* on page 500 of this issue.

► Establishment of the Carolina Subsection of the Southeast Section, upon submission of proposed bylaws and list of officers, was approved.

► The proposed Bylaws of the Council of Education of AIME were approved with the following revision: *Article II* was revised by transposing the last two lines of **Section le** to the end of the second line of *Article II*, following "AIME" and preceding **a**.

► Revised bylaws of the Student Chapter at Massachusetts Institute of Technology were approved. There is no longer a Mining Student Chapter at MIT; the present Student Chapter is known as The MIT Metallurgical Society.

► Revised Bylaws of the Industrial Minerals Division were approved by the Directors of the Society of Mining Engineers and were published in the January 1958 issue of MINING ENGINEERING. Certain changes had been suggested since that publication. The AIME Board of Directors approved the revised Bylaws, subject to reapproval by the Board of Directors of the Society of Mining Engineers.

► Revised Bylaws of the Minerals Beneficiation Division were published in the January 1958 issue of MINING ENGINEERING and approved by the AIME Board of Directors, subject to the approval of the Board of Directors of the Society of Mining Engineers.

► Note was taken of the formation of the CIM-AIME Edmonton Petroleum Engineering Section of the Petroleum and Natural Gas Division of CIM. This section in the future is entitled to an AIME Section Delegate.

► Kenneth E. Hill was appointed to succeed Lyman H. Hart, resigned, as one of three AIME representatives on the EJC Engineering Manpower Commission.

► AIME President Augustus B. Kinzel will be the official Institute delegate to the Golden Jubilee of AIChE in Philadelphia, June 22 to 27. AIME will present a Scroll of Greetings at a Convention on Wednesday morning of that week, at which time academic dress will be worn.

► The following meetings will be held by the AIME Board of Directors: June 16, New York; September 19, Salt Lake City; November 17, New York; February 15, 1959, San Francisco; and Feb. 17, 1959, San Francisco. There will be a meeting of the Executive Committee on December 15 in New York.

► The following resolutions, passed by the Council of Section Delegates of AIME on February 15, were presented to the AIME Board on February 18:

1) *Resolved*, that the Council of Section Delegates recommend to the Board of Directors of AIME consideration of a campaign to develop interest in the Earth Sciences and Mineral Industries among students in junior high schools.

ACTION: President Kinzel appointed Douglas Ragland, Chairman; W. R. Hibbard, Jr.; and A. M. Gaudin as a committee to study this subject and to report back to the AIME Board of Directors.

2) *Resolved*, that the Council of Section Delegates recommend that the Board of Directors of AIME appoint a committee to investigate the adequacy of activities in connection with the translation and distribution of foreign literature in areas related to the Mineral Industries and to recommend to the Board of Directors what program, if any, should be adopted.

ACTION: President Kinzel appointed a committee consisting of John Chipman, Chairman; J. C. Calhoun, Jr.; and J. R. Vanderwilt, Jr., to study the matter and report back to the AIME Board of Directors.

3) *Resolved*, that the Council of Section Delegates advise the Board of Directors of AIME that by a vote of 25 for and 23 against, with 18 Delegates (present) not voting, the Council of Section Delegates determined that Local Sections need an annual rebate from the AIME of $1 per member to permit Local Sections to carry out activities unique at the Local Section level.

ACTION: The matter was discussed with particular reference to the AIME Budget. It was referred to Dr. Gillson, Chairman of the Committee on Local Section Affairs, for a recommendation from his group.

Minnesota Symposium, Section Meeting, Have Record Attendance at Three-Day January Event

AIME Minnesota Group Holds Annual Meeting

With cooperation from the weather which was unseasonably mild with almost no snow, the Minnesota Section's Annual Meeting broke all attendance records with about 623 members and guests. The meeting was held in Duluth on January 13 and was followed by the two-day Annual Mining Symposium of the University of Minnesota.

John D. Boentje, Jr., vice president and general manager of Pacific Isle Mining Co., Hibbing, Minn., opened the proceedings as Section chairman. He made a brief welcoming address and thanked the various committeemen responsible for the meeting. After presentation of the secretary-treasurer's report by Norman A. Moberg, Oliver Iron Mining Div., U. S. Steel Corp., Duluth, Fred D. DeVaney, Pickands Mather & Co. read the report of the nominating committee and new Section officers were elected.

Newly elected Section officers for 1958 are: Stephen E. Erickson, chairman; Robert J. Linney, first vice chairman; Ralph W. Marsden, second vice chairman; R. W. Livingston, third vice chairman; and Robert L. Bennett, secretary-treasurer.

The rest of the meeting was devoted to morning and afternoon technical sessions. Grover J. Holt, AIME Past-President, attended and spoke briefly about his experiences as head of the Institute and on AIME activities.

One of the highlights of the meeting was the luncheon at which Laurence M. Gould, president of Carleton College, Northfield, Minn., was the guest speaker. Dr. Gould, who is head of American IGY operations in Antarctica, discussed *The International Geophysical Year in Antarctica*, and gave some little known facets of the U. S. scientific and educational programs as related to neighboring countries.

Two papers were presented at the morning technical session. Martin J. Hughes, manager of the Eagle Mountain mine of Kaiser Steel Corp., described this iron mining and beneficiation operation in Southern California. The second speaker, J. R. VanPelt, president of Michigan College of Mining and Technology, outlined current trends and views on *Engineering Education.*

Several papers were presented at the afternoon technical session. Dwight C. Brown, Jones & Laughlin Steel Corp., discussed *Direct Reduction Processes.* Peter Apostolakos, DM & IR Railroad, gave some aspects of *The Science of Human Relations.* The closing paper by A. E. Moss, Iron Ore Co. of Canada, was on *The Iron Ore Deposits of Labrador and Quebec.*

Major General L. J. Sverdrup, USAR, was the principal speaker at the annual banquet. He showed a film and related some of the engineering problems in airfield construction in the Southwest Pacific during World War II. His talk was entitled *Engineering in the Stone Age.* Warren S. Moore, W. S. Moore Co., Duluth, was toastmaster for the occasion.

Special event of the banquet was the presentation of the University of Minnesota Outstanding Achievement Award to Kenneth Duncan of Duluth. Mr. Duncan, retired general manager of Pickands Mather & Co., has been active in civic projects in Duluth since his retirement.

Minnesota Symposium On Mining Held In Duluth

Immediately following the Minnesota Section annual meeting, the annual Mining Symposium of the University of Minnesota was held at the Norshor Theatre in Duluth, January 14 and 15.

Preparation and Storage of Iron Ore Materials was the general theme of this year's symposium and the proceedings were opened by J. M. Nolte of the University's General Extension Div. A few words of welcome were also made by S. R. B. Cooke, head of the School of Mines and Metallurgy at the University.

A total of three sessions were held during the three-day meeting. The first was devoted to *Scrubbing of Ore Materials.* Papers and their authors were: *The History and Theory of Scrubbing* by R. C. Ferguson; *Scrubbing and Examples from Iron Ore Fields* by R. C. Engstrom; *Heavy Media Feed Preparation by Other than Rotary Scrubbers* by Don Kimball; and *The Scrubbing of Painty Iron Ores* by George Glumac.

The afternoon session on January 14 was devoted to *Screening of Iron Ores* and consisted of the following papers: *Problems in and Application of Fine Screening* by Chelsea R. Phillips; *Experience with Dutch State Mine Screens, Holman-Cliffs Cyclone Plant* by James Stukel; and *Experience with Derrick Screens* by Wm. R. Derrick. The last two papers on Tuesday were on the design and preparation of tailings sites. Charles Grant, J&L, outlined current methods and practices on the iron ranges for containing plant tailings. In comparison, John Grant, The Anaconda Co., pointed up the special problems encountered in the Butte and other western districts in tailings disposal.

The final session of the Symposium on Wednesday morning, January 15, dealt with *Surging and Stockpiling.* Among the papers presented were: *Surging—General Considerations* by Richard A. Derby; an outline of design considerations for surge piles and structures by Abe Mathews; *Stockpiling—General Considerations* by Wm. F. McDermott; *Application of the Cableway to Iron Ore Stockpiling and Reclaiming* by Wm. Lanctot and Alex Paton; and *Surge Pile Installations of IOCC at Seven Is.* by R. W. Kirkland.

Attendance at the Symposium was over 600. The next annual Mining Symposium is scheduled for January 1959.

Moa Bay Mining Sets Up Education Award Plan

Moa Bay Mining Co. of Cuba, a subsidiary of Freeport Sulphur Co., will launch a new type of international scholarship plan that not only pays the exchange student's expenses but also contributes to the school he attends. Announcement of the plan was made recently by Kenneth Holland, president of the Inst. of International Education.

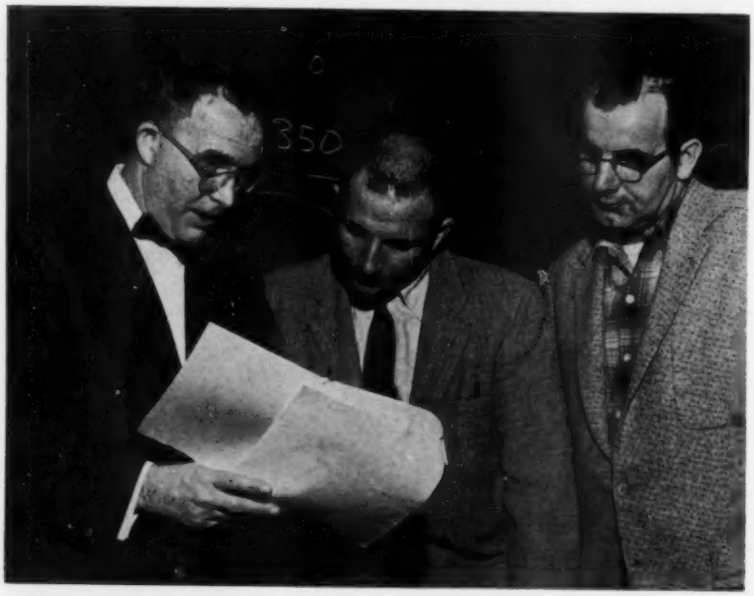
The fellowships, to be administered by the Institute, will pay the expenses up to $3500 each for four Cuban students to study in U. S. schools next year. In addition to paying tuition and other fees, the company will give a $500 grant to each of the four universities attended by the Cuban exchange students.

Three of the fellowships will be open to students in the fields of mining, metallurgy, geology, chemical engineering, the physical sciences, and other related fields. The fourth will be given in the field of atomic energy as applied to medicine.

Langbourne M. Williams, president of Freeport Sulphur, hopes that the program will help spur Cuba's economic development, particularly the development of its mineral resources. The plan is also a recognition of the need to support U. S. educational institutions, especially those which have been fostering international understanding through the exchange of students.

Around the Sections

• John S. Rinehart gave two talks to the **Mines Student Chapter,** Colorado School of Mines, Golden, on February 6 and 7. The assistant director of the Smithsonian Astrophysical Observatory, Cambridge, Mass., Dr. Rhinehart spoke on *Earth Satellites* and the failures effected on materials under explosive loads. "The epoch-making business of launching satellites corresponds to the world's greatest events—making these, indeed, historic times," he said. In his first lecture on February 6, Dr. Rinehart described the work of the Smithsonian Observatory in tracking satellites, Russian and the U. S. three. He also touched upon the role of the satellites in the International Geophysical Year. During his visit to the School of Mines, he stressed the need for men trained in the mineral field.



Shown talking to members of the Mines Student Chapter at the Colorado School of Mines is Dr. John S. Rinehart, left. Intent upon satellite orbit charts are George Beattie, center, a mining engineering junior from New Jersey and president of the Student Chapter, and Donald O. Rausch, instructor in mining at the School of Mines.

Members of the AIME Student Chapter at the School of Mines, West Virginia University, Morgantown, have been awarded scholarships for the 1957-1958 school year. They are, standing, J. V. Lundell, E. L. Murphy, Jr., J. W. Barton, M. J. Hudak, Jr., D. W. Curry, J. S. Ferrell, T. W. Garges, H. A. Walker, and E. C. Ford. Seated are Edward Kebblish, R. J. Dado, W. F. Renn, H. E. Harman, and V. T. Caruso. Messrs. Lundell, Barton, and Renn are recipients of grants-in-aid by the North American Coal Corp.; Messrs. Murphy and Kebblish hold scholarships from Armco Steel Corp. Other winners and their scholarships are Mr. Hudak, AIME Central Appalachian Section; Mr. Harman, WAAIME; Messrs. Ferrell and Dado, Semet-Sovay Div., Allied Chemical & Dye Corp.; Messrs. Curry, Walker, and Garges, Traux-Traer Coal Co.; Mr. Ford, Magnolia Petroleum Co.; and Mr. Caruso, graduate research assistantship, West Virginia Engineering Experiment Station. Not shown are P. G. Meikle, holder of a Carbon Fuel Foundation scholarship; and M. L. Teasdale, recipient of a North American Coal Corp. grant-in-aid.

• **Yavapai Subsection,** Arizona Section, held its regular monthly cocktail hour-dinner meeting on February 14. William Lord, Jr., was the featured speaker and gave a talk on mining in Bolivia.

• The city of Spokane has launched a supervised *space science* program for high school students. Organized by a citizen's committee and the Washington Soc. of Professional Engineers, the program is receiving support from local chapters of professional organizations, among which is the **Spokane Subsection** of the Columbia Section. The program will be divided into six divisions: rockets, astronomy, earth science, electronic subatomics, and space environment study, each administered by a professional or technical expert.

• Russel Londell was the featured speaker at the January 14 dinner meeting of the **Morenci Subsection** of the Arizona Section. Mr. Londell, head of the science department at Eastern Arizona Junior College, discussed *In the Realm of the Atom.*

The Subsection met again on February 25 at the Longfellow Inn. The program consisted of a showing of International Nickel Co.'s film, *The Milling and Smelting of Sudbury Nickel Ores.*

• The **San Francisco Section** had as its honored guest on February 26 Robert Koenig, president of Cerro de Pasco Corp. At the meeting in the Engineers Club, Mr. Koenig discussed the geology and mining of Peru where Cerro de Pasco has extensive interests.

• **El Paso Section** met on February 12 at the Hotel Cortez. J. E. Douglas gave an illustrated talk on the Glenn Canyon dam on the Colorado River in Arizona.

• The **Washington, D. C., Section** met on March 4 at the Cosmos Club to hear T. Reed Scollon. Mr. Scollon, chief of the Div. of Bituminous Coal, U. S. Bureau of Mines, discussed the development of coal resources in Alaska. His talk was illustrated with slides.

Utah Section Stages Second Successful Miners' Review

The Utah Section and WAAIME combined resources on February 8 at the Newhouse Hotel to produce a second sellout—the Annual Miners' Review.

The program for the evening started with cocktails and dinner in a music, fun, and cabaret atmosphere. Feature event of the program was the Miners' Review, followed by dancing and visiting.

In the Review, the dancing and singing group were comprised of: Kathleen apRoberts, Ginnie Dean, Dottie Harrison, La Rie Matheson, G. E. apRoberts, Charlie Hilton, Norm McLeod, and Bob Seklemian.

The May Dancers, of whom three are shown here, were Betty Norden, Bonnie Osterstock, Jackie Rosenblatt, and Pat Stevens.

A thriller-diller was the Musical Tragedy. Cast included: Melissa Jager, the beautiful heroine; Lois Lawson, the slinky villainess; Kay Olson, the mean old villain; Bob Portmess, the handsome hero; Edna Walker, the gin swigging rich countess; and Win Seymour, rigor mortis. Pulling the strings behind stage were: Stan Mitchell, stage director; Tom Harrison and Alan Jager, stage hands; and Betty Michaelson, makeup. Piano accompanist was Nell "88 Keys" Ensign, and group singing was under the direction of Ed Engelmann.

Responsible for all this was the Committee—Glen Burt, producer; Hop Ensign, master of ceremonies; Markey Eakland, script girl; and Mary Thuli, music and dialogue.

MAY DANCERS— Mrs. Joseph A. Norden, Jr., Mrs. Victor L. Stevens, and Mrs. Robert W. Osterstock.

MUSICAL TRAGEDY—Mrs. Melissa Jager, Robert R. Portmess, Mrs. Theland A. Walker, Emmett K. Olson, Jr., and Mrs. Lois Lawson.

RIGOR MORTIS—Winton L. Seymour.

DOUBLE QUARTETTE—Mrs. Thomas R. Harrison, Robert Seklemian, Mrs. G. E. apRoberts, Charles C. Hilton, Mrs. Kenneth H. Matheson, Jr., G. E. apRoberts, P. H. Ensign, and Mrs. Robert L. Dean.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Personals


H. J. MAYER


M. EICHELBERGER

Herbert J. Mayer has been elected to a two-year term as director of Region 11 of Associated Equipment Distributors. Mr. Mayer is executive vice president of Western Machinery Co., San Francisco.

H. Z. Stuart, formerly resident manager of mines, has been named manager of exploration by Phelps Dodge Corp., New York. In addition to directing operations in this country, Mr. Stuart will be responsible for the direction of exploration activities of the Canadian subsidiary, Phelps Dodge Corp. of Canada Ltd., Toronto.

John T. Atkins is now chief mine engineer for Utah Mining Corp., Lucky Mc Mine, Gas Hills, Wyo. Mining operations at Gas Hills are conducted by open pit methods and the uranium ore is being stockpiled pending completion of a new mill. Mr. Atkins had been assistant geologist for Union Pacific Railroad Co., Oil Development Dept., Denver.

Clyde N. Garman, formerly metallurgical engineer with the AEC, is now mill superintendent for the Homestake-New Mexico Partners.

Arthur R. Kinkel, Jr., of the U. S. Geological Survey, has returned from a two-month study of base metal and iron deposits in Turkey. This study was made at the request of the International Cooperation Administration.

The following officers have been elected to serve the Utah Mining Assn. for 1958: **Charles A. Steen,** president of Utex Exploration Co. and vice president and director of Uranium Reduction Co., Moab, is president; **Oscar A. Glaeser,** vice president and general manager of western operations for U. S. Smelting, Refining and Mining Co., Salt Lake City, is first vice president; **Lockwood W. Ferris,** president of Bonneville Ltd., Salt Lake City, is second vice president; **A. G. Mackenzie,** re-elected vice president and consultant; **Miles P. Romney,** re-elected secretary-manager; and **Walter M. Horne,** re-elected assistant secretary-manager.

Martin Eichelberger, recently a geophysicist for Geophysical Service Inc., has joined Aero Service Corp., Philadelphia. He will be a representative of the company's Airborne Geophysics Div.

Samuel A. Scott, formerly project engineer for the Colorado School of Mines Research Foundation, Golden, Colo., has joined International Minerals & Chemical Corp., Chicago, as a mining engineer.

Donald M. Wagg has been appointed chief geophysicist for Rio Canadian Exploration Ltd., exploration division of Rio Tinto Mining Co. of Canada Ltd. Mr. Wagg will also retain his position as operational manager of Gresham Exploration Ltd., a Rio Tinto subsidiary.

The Construction and Mining Div. of the Harnischfeger Corp. announced the promotion of five men in a realignment of the division's sales force. **F. J. Hirner,** district manager, Chicago office, since 1954 and a member of the P&H sales force for 19 years, was appointed sales manager-electric excavators. **W. N. Ryan** will succeed Mr. Hirner as district manager in Chicago. **G. T. Raubach** is assistant sales manager of electric shovels. Other changes include: **R. B. Maxson,** promoted from excavator salesman to district manager at Buffalo, and **C. R. Morgan, Jr.,** appointed to the position of sales manager, soil stabilizers and Sierra loaders.

George H. Cain has been appointed assistant secretary of Cerro de Pasco Corp., succeeding **Joseph F. McGowan,** named controller of the concern.

Raymond G. Lindlof, formerly geological engineer with the AEC in Casper, Wyo., is now chief engineer and geologist for Federal Uranium Corp., Salt Lake City.

Adrain R. Fisher president and director, also became chief executive officer of Johns-Manville Corp. in a realignment of executive responsibilities following retirement of **Leslie M. Cassidy,** chairman. **Clinton B. Burnett** was elected executive vice president and was made a director of Johns-Manville Corp.

Errol M. Kennedy has replaced **Karl R. Fleischman** as inspector of mines for the Fiji Government, Suva, Fiji Islands.

William M. McGill is now engaged in consulting geological work in Charlottesville, Va. He had been associated for many years with the Virginia Geological Survey, and for the last ten years as state geologist of Virginia.

D. R. Harries is now associated with Iron Ore Co. of Canada, Schefferville, Que., Canada.

Robert J. Hendricks has been promoted from resident director and manager in Nicosia, Cyprus, of Cyprus Mines Corp. to vice president. He has returned to the concern's office in Los Angeles. Mr. Hendricks

Mining research at the University of Illinois was discussed at the annual meeting of the industry's Advisory Committee on Mining Engineering at the University of Illinois. Present were committee members and representatives from the University's Dept. of Mining and Metallurgical Engineering and from the Illinois State Geological Survey, located on the campus at Urbana-Champaign. Left to right, seated, are: *F. Earle Snarr,* Freeman Coal Mining Corp.; *Theron G. Gerow,* Chicago mining consultant; *William L. Everitt,* University of Illinois; *Henry C. Woods,* Sahara Coal Co. Inc.; *Thomas A. Read,* University of Illinois; *Rudolph G. Wuerker,* University of Illinois; *Marcus L. Thompson,* Illinois State Geological Survey; *Walter H. Voskuil,* University of Illinois and Illinois State Geological Survey. Left to right, standing, are: *Frederick D. White,* University of Illinois; *Norman Street,* University of Illinois; *Ross J. Martin,* University of Illinois Engineering Experiment Station; *John G. Frye,* Illinois State Geological Survey; *Gill Montgomery,* Minerva Oil Co.; *J. Arthur Bottomley,* Sahara Coal Co. Inc.; *George W. Wilson,* Illinois Mining Institute; *William R. Chedsey,* University of Illinois; and *George R. Eadie,* University of Illinois.

personals

continued

was honored in the Queen's New Year's list with a Commander of the Most Excellent Order of the British Empire (C.B.E.). **John D. Burgess** has succeeded Mr. Hendricks as resident director and manager of Cyprus Mines in Nicosia.

Gordon S. Soine has returned to Oglebay Norton Co., Montreal, Wis., as mine engineer after service in the U. S. Air Force as a pilot.

Charles M. Romanowitz, for many years associated with Yuba Consolidated Industries Inc., most recently as director of dredge sales, Yuba Manufacturing Div., has resigned to join Ellicott Machine Corp., Baltimore, as manager of the company's new San Francisco office.

Lindell E. Montgomery, a recent graduate of St. Louis University, is now computer for Geophysical Service Inc., working on a field crew in southern Louisiana. He had spent two months in Dallas computing seismic records for Aramco. The records were made in Rubal Kali, Arabia.

Stewart Carpenter, who had been construction engineer for Erie Mining Co., Hoyt Lakes, Minn., is now project manager-construction for Andes Copper Mining Co., El Salvador project, near Potrerillos, Chile.

Stephen Krickovic, formerly chief engineer for the Coal Div., Eastern Gas and Fuel Assoc., Pittsburgh, has been chosen to head the newly reorganized Headuarters Engineering Dept. The department has been divided into six sections: mine engineering, preparation, property, electrical, ventilation, and shaft sinking and repairs. **E. B. Taylor,** formerly assistant chief engineer, is in charge of the mine engineering section with the title of chief mining engineer.

Vincent N. Burnhart, general manager of the Longyear Co., Minneapolis, has been appointed vice president. A member of the concern since 1946 and a director since 1952, Mr. Burnhart will continue to coordinate all activities of E. J. Longyear Co., Canadian Longyear Ltd., and two overseas operations in Paris and The Hague.

Bernard J. McCauley has been named new projects representative at the West Orange, N. J., laboratory of Vitro Corp.

W. H. Loerpabel has retired as general manager of the mining department operations for the South America division of American Smelting & Refining Co. He had been with Asarco since 1922 and was general manager for South America since 1951.

George R. McGrath recently became vice president and assistant to the general manager of the White Pine Copper Co., White Pine, Mich. Mr. McGrath, who joined White Pine in 1955, was appointed plant comptroller in 1956 and elected a vice president in 1957.

E. A. Schroer, mill superintendent of the Chino Mines Div., Kennecott Copper Corp., retired in January, 35 years after he first joined the division. A graduate of Missouri School of Mines, he is a native of Missouri and obtained his first job in Butte, Mont., in 1915. Joining Chino as a mill sample foreman in 1922, Mr. Schroer has been with the company since that time, except for a brief period during shutdown at the Hurley operations. During his career he rose, successively, to general mill foreman, assistant mill superintendent, and in 1948 to mill superintendent.

Richard B. Nelson, formerly with Hidden Splendor Mining Co., Moab, Utah, has joined International Min-

erals & Chemicals Corp. as mine project engineer. He had also worked in Carlsbad, N. M., for Potash Co. of America.

G. L. Jordan has become assistant general manager of National Potash Co. Previously refinery superintendent for the company, he will continue in this capacity and will also be in charge of surface maintenance, carloading, and the laboratory in Carlsbad, N. M.

A new mining and milling department has been organized by Phillips Petroleum Co. in Bartlesville, Okla., for the uranium ore extraction and processing operations of the company. **T. M. Hipp** heads the new department as manager, **A. A. Ruoho** is general superintendent, and **R. W. Jenkins,** mines superintendent. Mr. Hipp will make his headquarters in Bartlesville and Messrs. Ruoho and Jenkins will establish headquarters at the firm's Ambrosia Lake uranium processing mill north of Grants, N. M.

Grand Junction, Colo., operations office of the AEC has been reorganized and two new units formed—Production Evaluation Div. and the Source Materials Procurement Div. These replace the former operating divisions—mining, exploration, and concentrate procurement.
Arthur E. Granger, area manager at the Salt Lake City office, is the new director of the Production Evaluation Div. and **David D. Baker,** formerly director of the Mining Div., is deputy director. Other appointments include **Geraldine E. Martin,** administrative officer; **John G. Barry,** chief of special projects branch; **Charles A. Rasor,** chief of ore reserves and production analysis branch; **Robert H. Toole,** chief of leasing and development branch; **Hobart E. Stocking,** chief of production services branch; **Millard L. Reyner,** acting chief of the Casper, Wyo., branch; **Eugene W. Grutt, Jr.,** acting chief of the Grants, N. M., branch; **Ernest E. Thurlow,** chief of the Denver branch; **Hiram B. Wood,** chief of the Plateau branch; and **Wayne Bills,** acting chief of the geophysical services branch. **Charles E. Tonroy,** formerly director of the Concentrate Procurement Div., is director of the new Source Materials Procurement Div. Other appointments include **John W. Barnes,** chief of the planning and evaluation branch; **Michael P. Mehan,** chief of the concentrate procurement branch; **Frank E. McGinley,** chief of the procurement services branch; and **Gilman C. Ritter,** chief of the ore procurement branch.

Lance H. Cooper, London, England, vice president of The International Nickel Co. of Canada Ltd., has been elected a director of the company. Mr. Cooper, who joined Mond Nickel Co. in 1926, has been chair-

personals

continued

man of Mond Nickel, Inco's United Kingdom affiliate.

Lewis A. Parsons, consulting engineer for Calaveras Cement Co., San Francisco, was recently honored at a luncheon in recognition of his retirement after 21 years on the Calaveras staff. Mr. Parsons, who joined the firm in 1937 as mining and research engineer and who became consulting engineer in 1944, was presented with a gold wristwatch. He is one of the originators of the new system of automatic kiln control placed in operation in Calaveras about four months ago.

Robert F. Thurrell, Jr., who had been executive director of IREX (international resources engineering and exploration group) is now vice president and general manager of Global Enterprises Corp. This firm of consulting geologists is actively engaged in exploration and leasing in Alaska in addition to its activities in this country.

Kenneth F. Tupper, president of Ewbank & Partners (Canada) Ltd., has been elected president of the Engineering Institute of Canada for the coming year.

F. X. Tartaron, formerly manager of ore research for Jones and Laughlin Steel Corp., is now professor of mining engineering at the University of Pittsburgh and consulting engineer for U. S. Steel Corp., Pittsburgh.

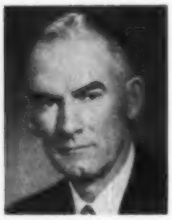
Van H. Smith, professional engineer, has joined Simard, Knight & Assoc., Toronto, as a mineral dressing consultant.

Roger A. Madsen has been transferred by Homestake Mining Co. from Lead, S. D., to Grants, N. M. At Grants, Mr. Madsen is maintenance superintendent.

M. G. Erasmus is now on the staff of the Amalgamated Bankel Areas Ltd., Tarkwa, Ghana. He had been associated with Frontino Gold Mines, Colombia.

B. A. Winn has joined the AEC as metallurgical engineer in the Concentrate Procurement Div. at Grand Junction, Colo. He had been connected with the Utah Construction Co. on the company's Korean project.

K. F. TUPPER



George A. Brecker has retired after 45 years of service with Reading Anthracite Co. where he was chief safety engineer. He had worked for the Philadelphia Reading Coal and Iron Co. and the Reading Anthracite Co. continuously with the exception of service during World War I.

Robert J. Hohne, who had been a first lieutenant in the U.S. Army Engineers, has joined Southern Counties Gas Co., Los Angeles, as assistant construction engineer. During his service in the Army, he was officer-in-charge of a first order triangulation arc which extended across Iran from Turkey to Pakistan. This program was under the supervision of the Army Map Service.

J. T. Mareta, a graduate of Colorado School of Mines, is working as a metallurgical engineer for the Inland Steel Co., East Chicago, Ind.

Earl H. Brown has retired from the Silver Bell Unit, American Smelting and Refining Co., Silver Bell, Ariz., and is residing at Grass Valley, Calif.

W. C. Lapple, formerly associated with Midwest Research Inst., Kansas City, has joined the staff of Oliver Iron Mining Div., U. S. Steel Corp., at the research laboratory in Duluth.

W. H. Leo has become manager of sales at the Sheffield Div., Armco Steel Corp. He had recently been on leave from the company while serving as director of the Iron and Steel Div., Business and Defense Services Administration, U. S. Dept. of Commerce.

Robert L. Kampen is plant foreman at the Sherman mine ore crushing and screening plant, Chisholm, Minn.

S. R. Zimmerman, Jr., has been appointed general manager of U. S. Asbestos-Grey Rock Div., Raybestos-Manhattan Inc., Manheim, Pa. He succeeds **O. H. Cilley**, who continues as vice president and director of the corporation.

James F. Connaughton has been elected president of Wheelabrator Corp., a subsidiary of Bell Aircraft Corp.

Louis J. Smith and **Edward T. Landgraff** have been elected vice president and sales manager of Chiksan Co. and Chiksan of Canada Ltd., and Chiksan Export Co., respectively.

Frank J. Wiebelt, recently retired as Bureau of Mines mining engineer, has been honored with a Commendable Service Award and Bronze Medal of the U. S. Dept. of the Interior. A veteran of 23½ years of Government service, Mr. Wiebelt now resides in Arizona.

John A. Hagan has been appointed assistant to the vice president–operations of U. S. Steel Corp. He had been general superintendent of U. S. Steel's Homestead District Works and in his new position will be responsible for short and long range planning for all the mining and manufacturing facilities of the corporation.

Roy V. Hersey, Coolidge, Ariz., has been appointed state mine and inspector to succeed the late Ed Massey. Mr. Hersey has had wide experience in mine safety work.

Donald R. Boynton is now special assistant to the sales manager of the mining and construction division of Gardner-Denver Co., at the firm's general offices in Quincy, Ill. Mr. Boynton, who has been a field engineer in the Toronto office, is a graduate of the University of Toronto with a degree in mining engineering.

personals

continued

Thomas T. Fleming, Jr., has been elected a director and vice president and general manager of Frank Samuel & Co. Inc., subsidiary of Haile Mines Inc. Mr. Fleming has been vice president and sales manager of the Philadelphia concern.

Raymond C. Moore, head of the department of geology at the University of Kansas, Lawrence, Kan., has been elected president of the Geological Soc. of America.

Jerome Strauss has retired as vice president of Vanadium Corp. of America after 30 years of service with the company. During his career of guiding research for Vanadium Corp. he was responsible for the initiation of the low-alloy high-strength plate and sheet steels as well as certain alloys which are now commonly referred to as the boron-treated steels. Patents have been issued in his name on both of these developments. Active in metallurgy for over 40 years, he was in 1953 the ASTM Gillett Memorial Lecturer and in 1955 the Burgess Memorial Lecturer, Washington Chapter of ASM. He received in the same year an honor award from Stevens Institute of Technology, his alma mater. Mr. Strauss plans to undertake a limited amount of consulting work both here and abroad and is at present in Europe in this connection.

R. E. Young has been named to the newly created position of special representative for the Electric Excavator Div. of Harnischfeger Corp., Milwaukee. Mr. Young, who has been with P&H for 41 years, has been associated with the heavy mining industry and more particularly with the open pit operations.

R. E. YOUNG

J. C. CLAY

Jeff C. Clay has been elected vice president of engineering of The Long Co., Oak Hill, W. Va., mining equipment manufacturer.

Paul Norman Clawson has joined the mining and exploration department of International Minerals & Chemical Corp., Chicago, as a geologist. He is a graduate of the University of Illinois and Montana State University with a master's degree in geology.

George P. Towle, has been named president of Sturtevant Mill Co. of Dorchester, Mass. He had been executive vice president and general manager.

Wayne Burnside is now sales manager of Brunner & Lay Inc., Philadelphia. He had been sales manager of Brunner & Lay Rock Bit and has had extensive experience in sales and pneumatic accessory tools and carbide drill bits service.

George E. Kruger is now a staff member of the Chase Manhattan Bank, New York, as mining geologist. He had been associated with Cerro de Pasco Corp. and Ventures Ltd. and has also done consulting work in mining geology in South America and New York.

W. BURNSIDE

G. E. KRUGER

Captain B. L. Lubelsky, USN, has assumed command of the large southern Indiana Naval Ordnance Station. He succeeds **Captain H. S. Harnly,** who has recently retired to civilian life. Captain Lubelsky, a graduate of Carnegie Institute of Technology, with an M.S. degree in explosive engineering and of the University of Illinois with a professional degree, engineer of mines, is an authority in explosives and blasting. He spent two years as explosives engineer with the Cardox Corp., seven years as explosives engineer with the Philadelphia Reading Coal and Iron Co., one year as consulting engineering manager with the Castle Shannon Coal Co. He has written numerous articles on explosives, mining, and blasting.

Joseph C. Abeles has been elected a director of Haile Mines Inc., New York. Mr. Abeles is director, vice president, and treasurer of Kawecki Chemical Co. of Boyertown, Pa.; and he is also a director of Hexagon Laboratories, New York, and Talco Engineering Co. of Hamden, Conn.

Maurice Deul, formerly a research geologist with the U. S. Geological Survey, has joined the staff of Bituminous Coal Research Inc. and has been assigned to the Pittsburgh laboratory, where he will devote his efforts to the field of earth sciences, primarily to chemical and mineralogical research and economic geology. **Robert L. Sankey** of the Pittsburgh staff has been promoted to publications manager.

M. DEUL

R. L. SANKEY

Andrew Fletcher, president, St. Joseph Lead Co., and AIME Past-President, was the recipient of an honorary Doctor of Laws Degree conferred upon him by Seton Hall University, South Orange, N. J., in December. The awarding of honorary degrees was part of a university convocation on industrial health. In the citation which accompanied Mr. Fletcher's degree mention was made of his efforts as chairman of the Industrial Foundation of America to improve industrial hygiene. It read in part, "His personal and professional interest in Industrial hygiene and his untiring zeal for the betterment of humanity make it a privilege to present Mr. Andrew Fletcher as a candidate for the degree of Doctors of Laws, honoris causa, with all the

rights and privileges appertaining thereto." Mr. Fletcher has also received an honorary Doctor of Engineering degree from the University of Missouri.

Ta Liang, a professor at Cornell University, has joined Aero Service Corp., Philadelphia. Dr. Liang will act as consultant to assist in air photo interpretation.

Morris Mielke has become assistant general superintendent at Oliver Iron Mining Div., U. S. Steel Corp., Eastern District Operations on the Mesabi Iron Range.

Several personnel changes recently took place at MacIntyre Porcupine Mines Ltd., Toronto. **John D. Barrington** is now president and managing director. **S. M. Wedd** has been elected a director and **M. L. Urquhart** has been named vice president-operations.

Two appointments have been made by Crucible Steel Co. of America for coal mining operations. **A. V. Faull** is superintendent of a coal mine at Hugheston, W. Va., a mine recently acquired by the Coal Mining Dept. of Crucible. **Frank A. Burns** is acting superintendent of the coal mine, Crucible, Pa.

John J. Thiessen is now mines evaluator for the Yuba Mining Div. of Yuba Consolidated Industries Inc. in California. He had been superintendent of exploration for the Glidden Co., Baltimore. While with Glidden he had made an initial evaluation of an ilmenite deposit in Northern Florida. Prior to that time, Mr. Thiessen had been with Union Carbide Ore Co. and had participated in mineral reconnaissance in Africa, South America, and the southern states of this country.

W. D. Michaely, sales engineer for Western Machinery Co., has become district representative with headquarters in Phoenix, Ariz.

Robert M. Jones, a recent graduate of Colorado School of Mines, is working for Orinoco Mining Co. in Estado Bolivar, Venezuela. He is presently connected with the ore grading and mine planning department.

W. Lee Heidenreich, an exploration and examination engineer for the U. S. Bureau of Mines, recently completed a drilling project of large low-grade nickel and cobalt laterite-serpentine deposits in western Puerto Rico. He has returned to Rolla, Mo., to write a report on the project which he hopes will eventually be published.

Ralph W. Cote, Jr., who had been shift boss for the Bunker Hill Co., Kellogg, Idaho, has joined Union Carbide Nuclear Co., Slickrock, Colo., as production engineer.

Robert C. Bates is now connected with the U. S. Bureau of Mines, Denver, as mining engineer in the Health and Safety Activity area. Mr. Bates had been mining engineer and geologist with Balboa Mining and Development Co., Grand Junction, Colo.

Grover J. Duff retired in January from the Eagle-Picher Co. after almost 25 years of service. After periods with the Highland Boy Mine, North Lily Mine, and the Vipont Mine, properties in Utah of the Anaconda Copper Co., and at Idaho-Maryland Mine and the Oceanic Quicksilver Mine in California, and the Black Bear Mine in Colorado, he joined Eagle-Picher in 1934 as superintendent of operations at the Montana Mine, Ruby, Ariz. Following this assignment, Mr. Duff became manager of operations at Marion, Ky., before returning to Arizona as western manager. During the time he was western manager his chief responsibilities were the San Xavier Mine and the Sahuarita Mill near Tucson. On January 31 Mr. Duff, who will con-

tinue to live in Tucson, completed a 5-year appointment as member of the Arizona Highway Commission, serving for 12 months as chairman.

G. J. DUFF

Rodney C. Foster, a recent graduate of Montana School of Mines, is now office engineer for Ingersoll-Rand Co. in Salt Lake City.

Ellis B. Herrington, Jr., who had been a student, is now mining geologist for American Smelting and Refining Co. in Mexico.

Keith N. Meador has moved the headquarters of his consulting business from Reno to Fallon, Nev.

Charles R. Pace, Jr., who has been mine engineer for United Clay Mines Corp., has been promoted to assistant superintendent of the Hawthorne, Fla., operation of the company. He had been located at the Sandersville, Ga., operation.

OBITUARIES

James Latimer Bruce

An Appreciation By
George D. Dubb

Some associates whose qualities are taken for granted are not appreciated until suddenly we are bereft of their company.

Not such a man was Jim Bruce. Straight thinker, keen observer, peerless engineer, able executive, his advice always virile, was sought by all, was readily and cheerfully given, and was usually the solution. Sympathetic, friendly, generous, painstaking, patient, even-tempered, he was ready always to be of assistance to younger members of his profession. Many prominent engineers received much of their training while associated with him.

In groups where, because of his years and his experience, he was the elder statesman—yes, the patriarch—his outlook was usually the youngest, the most forward looking, the one which kept before it always with single purpose the goal towards which the group should be directing its efforts. Never did he lose faith in the mineral industries.

His counsel, his friendliness, his frankness, his humility, the twinkle in his eye, will all be sorely missed by his associates. The void will not be readily filled.

Erle Victor Daveler

Memorial Resolution Prepared by Andrew Fletcher

WHEREAS, with the death of Erle Victor Daveler on November 11, 1957, the Nation lost one of its renowned engineers, and the Institute a most devoted member, who had served with distinction as a Director from 1929 to 1951, as Vice-President from 1940 to 1949, and for many years as Chairman of the Investments Committee, helping to pilot the Institute through several critical years, and in 1949 became Honorary Member, and

WHEREAS, during his lifetime, he served as an officer of many mining and allied corporations, developing some of the Nation's important resources in gold, copper, lead, zinc, iron and cement, and

WHEREAS, throughout his entire career he was an inspiration to his associates by virtue of the high standard he set for himself.

THEREFORE, BE IT RESOLVED, that the American Institute of Mining, Metallurgical, and Petroleum Engineers, express its deep sorrow over the loss of this distinguished member and friend; and

BE IT FURTHER RESOLVED, that this Resolution be spread upon the Minutes of this meeting and a copy be sent to Mrs. Daveler.

February 16, 1958

James L. Bruce (Legion of Honor Member 1903), vice president, consulting engineer, and director of Cyprus Mines Corp., passed away suddenly on Feb. 6, 1958, at the age of 77. He is survived by his widow, Leah Hills Bruce; 4 children; 11 grandchildren; and 2 sisters.

Born in Dublin, Ireland, his early schooling was obtained largely at home near Haliburton, Ont., Canada, and then in school at Woodstock, Ont., until at 13 he joined his father in Denver. He entered Colorado School of Mines in 1896, and was graduated in 1901 after having taken a year's leave between sophomore and junior years to earn sufficient income to enable completion of his course.

Until 1904 he worked for various firms at Cripple Creek as surveyor, assayer, draftsman, and engineer. In 1904 he went to southeast Missouri to work for the Federal Lead Co., managed at that time by J. R. Finlay. His work continued here as surveyor, chief engineer, and general mine foreman until 1907 when he went to Joplin for various companies and finally with Continental Zinc Co. as manager.

In 1913 he became associated with Butte and Superior Co. which D. C. Jackling was developing. During this connection Mr. Bruce took a prominent part in two important lawsuits, one with the Minerals Separation Co. in connection with flotation patents and the other with the Clark Co. in apex litigation. Butte at this time was also the scene of serious labor difficulties which added to his rapidly increasing store of knowledge.

In September 1919 Mr. Bruce became manager of the Davis-Daly Copper Co. at Butte where he remained until 1924.

From 1924 to 1925 he was a consulting engineer in Salt Lake City following which he began his long association with Cyprus Mines Corp.; to 1939 as manager and resident director in Cyprus and thereafter as vice president, consulting engineer, and director in the U. S. During this time, among many other duties, he was responsible for the design of the sulfuric acid autoxidation plant which was completed in Cyprus in late 1951, and was closely connected with the affairs of Pima Mining Co. and Marcona Mining Co.

Mr. Bruce was an AIME Legion of Honor Member, having joined the Institute in 1903. He was a member of the American Mining Congress and of the Engineers Club of Los Angeles. During World War II he served as consulting engineer for the War Production Board. Prior to Pearl Harbor he recommended to OPA the Premium Price Plan which was subsequently used for stimulating zinc production.

Mr. Bruce was the author of numerous articles on Cyprus published by AIME and the Colorado School of Mines Alumni Bulletin. With C. P. Manglis and D. M. Creveling, he wrote *Antiquities in the Mines of Cyprus* which formed appendix V, vol. III, 1937, of *The Swedish Cyprus Expedition*.

In 1949 at the 75th anniversary convocation of the Colorado School of Mines, Mr. Bruce was awarded the Distinguished Achievement Medal.—*G. G. Dubb*

Edgar C. Agabeg (Legion of Honor Member 1905) died recently. He had been a consulting mining engineer in Asansol, Bengal, India. Born in Calcutta in 1862, Mr. Agabeg received his education in that city. He had been associated with mining and coal concerns in his native country.

Charles H. Bowen (Member 1935) died in Monrovia, Calif., on January 4. Mr. Bowen was assistant chief, Plateau Branch, in the Grand Junction operations office (raw materials) of the Atomic Energy Commission. Born in Creston, S. D., in 1911, he graduated from South Dakota School of Mines with a B.S. in geology and from Ohio State University with M.S. and Ph.D. degrees. He began his professional career at the Kettleman lead refinery of American Smelting & Refining Co., Omaha, Neb. Other jobs with Asarco took him to Barber, N. J., as assistant general refinery foreman, later as technical advisor in the copper casting department; and then to Bonanza, Nicaragua, where he was, successively, assayer, junior engineer, and chief engineer for Neptune Gold Mining Co., an Asarco subsidiary. After a stint in the U. S. Army Corps of Engineers in World War II, Dr. Bowen joined the Ohio Div. of the Geological Survey as assistant geologist. Returning to Ohio State University to study for his advanced degrees, he was also a part-time mineral resources engineer at the Ohio State University Engineering Experiment Station and was also a research assistant professor. Dr. Bowen became associated with the AEC in 1955 as assistant district geologist, later being promoted to district geologist in Grand Junction. He became assistant chief of the Plateau Branch just before his death. The author of numerous contributions to the technical literature, Dr. Bowen was a member of the Geological Soc. of America, the Soc. of Economic Paleontologists and Mineralogists, the Grand Junction Geological Soc., and the Ohio Academy of Science. Friends have established a memorial fund, deposited in the First National Bank of Grand Junction, to assist in the education of his children, Charles Randall and Keith.

James D. Francis (Member 1951), former president and board chairman of Island Creek Coal Co., died on January 8. A native of Pikeville, Ky., in 1884, he graduated from the University of Virginia Law School in 1908. Mr. Francis began law practice in Pikeville in that year and his legal corporation work led to his association with Island Creek and Pond Creek Coal Cos. as vice president. In 1934 Mr. Francis became president of Island Creek and chairman of the board in 1949. He retired in 1952 but remained a director. Mr. Francis spent the major part of his career in the coal industry in building the companies with which he was associated—by acquiring and developing properties, arranging

financing, acting in a legal capacity in connection with transportation matters and anti-trust actions. Chairman of the mineral law section of the American Bar Assn. in 1931, he presented a paper on anti-trust laws affecting coal sales. Mr. Francis was a member of numerous professional organizations, among which were American Mining Congress, National Coal Assn., Southern States Industrial Council, and West Virginia Coal Assn. He was the recipient in 1951 of the AIME Charles F. Rand Gold Medal.

F. V. C. Hewett (Member 1952) died on Dec. 27, 1957, in Toronto. Mr. Hewett was president and managing director of McIntyre Porcupine Mines Ltd., Toronto. A native of England, he graduated from the University of Toronto in 1933 with a B.A.Sc. in mining engineering. His early association with the mining industry was as field editor for *The Northern Miner*, a post he held for seven years. Soon after the outbreak of World War II in 1939 he went into war work, later becoming deputy metals controller for the Canadian government. During the Korean War, he served as director of the nonferrous metals division of the Canadian Dept. of Defense Production. After a period in private consulting practice, Mr. Hewett assumed his posts with McIntyre Porcupine in 1956. He had directed McIntyre's participations in Algoma Steel Corp. and Ventures Ltd. Mr. Hewett was a director of several mining companies and had been active in CIM and the Canadian Metal Mining Assn. Most recently he was a member of the committee for the Sixth Commonwealth Mining and Metallurgical Congress held in the fall of 1957.

Charles Hoyle (Member 1916) died on Dec. 28, 1957 at the age of 78. A consulting engineer engaged in examination work and geological surveys and an advisor on Mexican mining matters, he was born in St. Louis and attended the School of Mines and Metallurgy at the University of Missouri. After brief periods during summer vacations with minerals firms in the U. S., Mr. Hoyle first went to Mexico in 1902 where he did assay work for the Hammond staff of the Guggenheim Exploration Co. After several jobs for the company in Mexico and the U. S., he became manager of the Esperanza Mining Co., a Guggenheim property in Mexico. He remained with the company after it was acquired by English interests and later by Mexican. Leaving in 1924, he then went into private consulting practice in Mexico, working for various companies as well as doing numerous examinations and geologic surveys.

Peyton L. Mitchell (Member 1953) died on Oct. 26, 1957. A graduate of

the University of Kentucky in 1942 with a B.S.C.E. degree, he was born in Lothair, Ky., in 1920. After service in the U. S. Army during World War II, Mr. Mitchell joined Knott Coal Corp., Anco, Ky. With that company he was, successively, assistant general manager, superintendent, treasurer, and general manager.

In Memoriam

Mrs. Caroline Klein, mother of Mrs. E. Harvey Sharp (Irene Klein) of the AIME staff and a member of WAAIME, passed away Feb. 15, 1958. Mrs. Klein was known to many AIME members as Mother Klein.

Necrology

Date Elected	Name	Date of Death
1903	James L. Bruce Legion of Honor	Feb. 6, 1958
1956	Leon L. Larche, Jr.	Dec. 30, 1957
1946	Mark C. Malamphy	Mar. 18, 1957
1940	Joseph V. Mather	May 15, 1957
1912	James D. Mooney	Sept. 21, 1957
1924	John J. Oberbillig	January 1958
1938	Karl F. Peterson	Feb. 8, 1958
1952	Frederick W. Seyfarth	Unknown

MEMBERSHIP

Proposed for Membership
Society of Mining Engineers of AIME

Total AIME membership on Feb. 28, 1958, was 29, 911; in addition 2,959 Student Members were enrolled.

ADMISSIONS COMMITTEE

E. H. Crabtree, Jr., Chairman; Frank Ayer, Jack Bonardi, Edward G. Fox, J. A. Hagy, F. W. McQuiston, Jr.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

Ottar Brekke, Bjornevatn, Norway
Willis H. Brett, Aurora, Ill.
Leslie W. Camp, Provo, Utah
Walter W. J. Croze, Jr., Duluth
Henry E. Duchnowski, Johnstown, Pa.
Albert R. Eckel, Clover, S. C.
Edilbert Escande, Kroh, Upper Perak, Malaya
George E. Field, Baguio City, P. I.
William T. Forsyth, Center Valley, Pa.
William F. Hackett, Wauwatosa, Wis.
Cecil T. Hambley, Toronto
Haydn Hammonds, White Haven, Pa.
Robert C. Heath, Fort Collins, Colo.
Carl E. Hultquist, Johannesburg, South Africa
Harold C. Jeppsen, Weatherly, Pa.
Hidemasa Kubo, Tokyo, Japan
James M. Neilson, Houghton, Mich.
Hiroshi Ohya, Tokyo, Japan
John E. Pearson, Grants, N. M.
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Harry E. Redenbaugh, Pittsburgh
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Jens O. Rhude, Hibbing, Minn.
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Harry Charles Hersker, White Haven, Pa.
Carl R. Swenson, Tucson, Ariz.
Thomas P. Wollenzien, St. Paul

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John H. Bassarear, Tahawus, N. Y.
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Wallace T. Dow, Hanover, N. M.
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James W. Hager, Carlsbad, N. M.
Robert M. Hagerman, Mt. Tabor, N. J.
Howard L. Hartman, University Park, Pa.
L. E. Lynd, North Plainfield, N. J.
Keith G. Papke, Tucson, Ariz.
John C. Palombo, Dover, N. J.
Howard H. Rice, Silver Bell, Ariz.
R. M. Stewart, Climax, Colo.

REINSTATEMENTS

Members

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William L. Kendrick, Henderson, Nev.

REINSTATEMENT—CHANGE OF STATUS

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J. Grant Goodwin, Mill Valley, Calif.
Pierrepont A. Meyer, Jr., Lakewood, Colo.

Student to Member

Frank M. Antonioli, Butte, Mont.
L. D. Barry, La Sal, Utah
Harold J. Christy, Wharton, N. J.
Barton M. Collinge, New York
Douglas V. Watrous, Idaho Springs, Colo.

Student to Associate

Joseph E. Shaw, Butte, Mont.

Student to Junior

Daniel G. Parrillo, Trenton, N. J.

See pages 521, 522, 523

Professional Services

Appraisals
Assayers
Chemists
Construction
Consulting
Drilling

Geophysicists
Drilling
Management
Metallurgical
Reports
Valuations

Space limited to AIME members or to companies that have at least one member in their staffs. One inch, $50 per year; half inch, $30 payable in advance.

Continued on Page 522

Appraisals
Assayers
Chemists
Construction
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Continued from Page 521

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